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Recovery of rare earth elements

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ABSTRACT

A process for recovering rare earth elements from a composite ore containing rare earth elements that includes a monazite group mineral and an apatite mineral, includes pre-leaching the composite ore with an acid so as to substantially dissolve the apatite mineral into the leach liquor and precipitating rare earth elements from the pre-leach liquor. The residue of the pre-leaching step is subjected to an acid bake treatment, followed by a water leach, to produce a water leach liquor rich in rare earth elements. Impurities including thorium and iron are separated from the water leach liquor by introducing a neutralising additive to the water leach liquor rich in rare earth elements, and rare earth elements are precipitated from the post-neutralisation liquor.

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COMPLETE SPECIFICATION
STANDARD PATENT

Invention Title: **Recovery of rare earth elements**

The following statement is a full description of this invention, including the best method of performing it known to us:

Recovery of Rare Earth Elements

Field of the invention

This invention relates generally to the recovery of rare earth elements from ores containing the elements. More particularly, the invention is concerned with recovering rare earth elements from a composite ore containing rare earth elements that includes a monazite group mineral and an apatite mineral.

Background of the invention

Rare earth elements are of increasing economic importance as the highly specialised properties of individual elements find applications in a wide range of pervasive products. For example, lanthanum is a component of catalysts employed in oil refineries, neodymium is a component of solid state lasers, neodymium magnets are widely employed in headphones, hard disk drives, and DVD and CD-ROM drives, and europium and terbium are dopants in primary colour phosphors for plasma and LCD screens.

At present, the principal source of most rare earth elements is China, and the principal deposits are in the Sichuan and Baotou regions.

With limited prospects for expanded production, and diversion of an increasing proportion to domestic customers, China is unlikely to be able to meet the expected increase in world demand for rare earth elements over the coming years. Since the gap will not be met by traditional sources outside China, such as monazite deposits for certain of the elements, there is a growing requirement to tap non-traditional ores containing rare earth elements. One of these is a composite ore containing rare earth elements that includes a monazite group mineral and an apatite mineral. Apatite is a phosphate of calcium, while a "monazite group mineral" is a composite monoclinic phosphate of rare earth elements and several other substituents.

When a composite ore of this structure has been subjected to conventional or other known processing regimes, it has proven difficult to convert the ore to an economically useful deposit. High thorium content in both principal components of the ore have proven a hurdle to economic conversion of the phosphate base of the apatite to a "clean" fertiliser product, while the uranium content associated with the thorium cannot be economically extracted in sufficiently pure form.

Primary treatments that were attempted included a nitric acid leach, a hydrochloric acid leach, a sulphuric acid bake/water leach and a caustic crack. It was found that direct treatment of the ore resulted in either low rare earth recovery, or poor leach selectivity for rare earths over calcium and phosphorous.

It is an object of the invention to provide a process for recovering rare earth elements from composite ores containing rare earth elements that include a monazite group mineral and an apatite mineral, preferably in a manner that also permits economic extraction of the phosphate values of the ore and removal of thorium to an extent that can meet end-product specifications.

Disclosure of the invention

The invention provides, in a first aspect, a process for recovering rare earth elements from a composite ore containing rare earth elements that includes a monazite group mineral and an apatite mineral, including:

pre-leaching the composite ore with an acid so as to substantially dissolve the apatite mineral into the leach liquor;

precipitating rare earth elements from the pre-leach liquor;

subjecting the residue of the pre-leaching step to an acid bake treatment, followed by a water leach, to produce a water leach liquor rich in rare earth elements;

separating impurities including thorium and iron from the water leach liquor by introducing a neutralising additive to said water leach liquor rich in rare earth elements; and precipitating rare earth elements from the post-neutralisation liquor.

It is found that the acid pre-leach is effective in achieving sufficient dissolution of the calcium phosphate lattice of the apatite to allow rare earth elements to report to both the leach liquor and the solid residue.

The acid employed for the pre-leach is preferably nitric or hydrochloric acid, more preferably the latter.

Preferably, rare earth elements are precipitated from the pre-leach liquor as phosphates.

Precipitation of rare earth elements as phosphates from the pre-leach liquor is conveniently by addition of ammonia or hydrated lime. Advantageously, precipitation conditions are selected to recover, with the rare earth elements and in a form separable subsequently, a major proportion of thorium present in the pre-leach liquor from the composite ore.

Preferably, especially where the apatite is a fluorapatite, a fluoride component of the pre-leach liquor is removed, e.g. by addition of a sodium salt to cause precipitation of Na_2SiF_6. Silicon is typically sufficiently present in the composite ore to allow this precipitation. The sodium salt conveniently matches the acid employed for the pre-leach, for example NaCl in the case of a hydrochloric acid pre-leach or $NaNO_3$ with a nitric acid pre-leach. The sodium salt is typically required to stoichiometric excess, e.g. 7 to 8 times. Recovery of the fluoride component is substantially more effective when the pre-leach acid was hydrochloric.

The acid bake is preferably carried out in a rotary kiln, and is preferably effected for between 1 and 3 hours, most preferably 1.5 to 2.5 hours.

The acid bake treatment is preferably a sulphuric acid bake, and typically at a temperature in the range 200-280°C, more preferably 210-250°C, most preferably about 230°C.

Preferably. the thorium and iron separation is effected by neutralisation with a suitable alkali or alkaline earth additive selected to minimise rare earth co-precipitation or absorption. A calcium compound is thought to be unsuitable for the latter purpose. A magnesium compound such as magnesium oxide or magnesium carbonate is satisfactory and may be preferred.

The pH of the liquor following addition of the neutralisation additive is preferably in the range 2 to 3.5, most preferably about 2.5. This pH is thought to be optimum for maximum thorium and iron removal whilst minimising rare earth loss to the solid residue of the neutralisation step.

It is found that a minimum Fe:P molar ratio of 1, preferably an Fe:P molar ratio substantially equal to 1, in the water leach liquor feeding the neutralisation is desirable in order to minimise rare earth loss to the solid residue by premature precipitation of rare earth phosphates. Hence, it is preferred if necessary to add Fe upstream of the neutralisation step, e.g. to the acid bake, in order to optimise thorium and iron removal with minimal rare earth element loss to the solid residue of the neutralisation step. Preferably, thorium removal is optimised to ensure an acceptable thorium level in the downstream rare earth end-product.

The precipitation of the rare earth elements from the post-neutralisation liquor is preferably by carbonate or double salt precipitation. The resultant composite rare earth carbonates or hydroxides may be leached to produce a composite rare earth chloride as an end-product with acceptable impurity levels. The leach may be particularly desirable to remove aluminium from the composite rare earth carbonate.

With carbonate precipitation, 110% stoichiometric addition of a carbonate is preferred.

In an embodiment, the rare earth phosphates precipitated from the pre-leach liquor are subjected to said acid bake together with the pre-leach residue. Alternatively, the rare earth phosphates precipitated from the pre-leach liquor are separately treated by an acid bake or by a caustic conversion process to permit recovery of rare earth elements therefrom.

Preferably, the pre-leach acid addition is about 110% stoichiometric with respect to calcium. As mentioned, apatite is a phosphate of calcium and calcium may also be a significant substituent in the monazite group mineral, for example where it is cheralite. This level of acid addition with respect to calcium optimises calcium removal by subsequent precipitation from the liquor, for example by crystallisation as calcium nitrate.

The initial concentration of the nitric acid pre-leach is preferably in the range 30 to 60wt%, more preferably in the range 40 to 50wt%. The initial concentration of the hydrochloric acid pre-leach is preferably in the range 10 to 40wt%, more preferably in the range 20 to 30wt%.

Examples

A composite ore from Nolans Bore in the Northern Territory of Australia was subjected to a series of processing tests according to the invention. Rare earths in the ore were present in cheralite, (LREE, Ca)(P,Si)O_4, which is a monazite group mineral, in association with fluorapatite, $Ca_5(PO_4)_3$ (F,OH). The main ore resource consists of about 82% w/w fluorapatite and 13% w/w cheralite. The remainder of the ore is about 3 to 4% clay with some calcite, dolomite, quartz and other minor minerals. Rare earths substitute for the calcium in the apatite structure. However, the bulk of the rare earths and thorium in the ore is contained in the cheralite, which is isomorphous with monazite. SEM examination suggested that approximately 67% of the rare earths are associated with the cheralite and 33% are associated with the apatite. Two assays of the composition of the ore, determined by XRF, are set out in Table 1.

Samples of the ore, riffled and ground to P_{80}=180 μm, were subjected respectively to nitric acid and hydrochloric acid pre-leach treatments. The details and results of these treatments are set out respectively in Table 2 and Table 3. In the nitric acid pre-leach (Table 2), Ca and P extractions of 97% and 94% respectively were obtained. Rare earth dissolution of 40% can be expected. A rare earth residue concentrate containing <3% Ca and P, and 11% rare earths was produced. Similar results were obtained by hydrochloric acid pre-leaching (Table 3).

In the example reported in Table 2, the initial HNO_3 concentration is 54wt%. Further calculations and experiments have indicated that the broadly preferred initial HNO_3 concentration is 30 to 60wt%, but it is more preferably in the range 40 to 50wt%. The corresponding figures for the hydrochloric acid pre-leach are 10 to 40wt% and 20 to 30wt%.

The pre-leach liquors were treated by addition of ammonia to successfully precipitate rare earth phosphates. The details of the treatments and the results are set out in Tables 4 and 5. Rare earth recoveries of 89% and 98% were obtained from the nitrate and chloride liquors respectively.

Four samples of solid residue from the hydrochloric acid pre-leach were subjected to a sulphuric acid bake in a rotary kiln, under standard conditions of 750 kg/lt H_2SO_4 addition and a temperature of 230°C for 2 hours. Rare earth recovery was consistently about 96%. It had previously been ascertained, in smaller sample tests in a static furnace that rare earth recovery fell from 90-92% at 230°C to less than 80% at 200°C, and was not improved by increasing the temperature from 250°C to 300°C. As earlier foreshadowed, an Fe:P ratio of about 1 is thought to be required for an optimum neutralisation step, so bakes of pre-leach residue at temperatures above 250°C are preferably avoided since they were found to reduce Fe dissolution.

In the smaller sample tests, a bake time of 2h was found to be optimum; at a bake time of 0.5h, rare earth recovery was significantly lower.

The solid residue of each acid bake test was water leached to produce a water leach liquor rich in rare earth elements. Waste solids were removed: these were found by SEM and XRD analysis to contain very little rare earth remnant and to consist mainly of quartz, alkali feldspar and remnant silicates, surrounded by clays and micas. The liquor was subjected to a standard solvent extraction procedure to remove uranium values, e.g. employing a tertiary amine as solvent.

Neutralisation of the water leach bake liquor was then carried out as a purification step to selectively precipitate thorium, iron and aluminium from the rare earths. This was effected at room temperature using 20wt% slurries of the neutralising additive, magnesia or magnesium carbonate. It was found that calcium carbonate and lime were less effective neutralising agents due to much higher rare earth losses, possibly due to co-precipitation/adsorption with gypsum due to the added calcium.

By testing, it was further found that a minimum Fe:P molar ratio of above 1, preferably about 1, was desirable in the water leach liquor feeding the neutralisation in order to prevent rare earth loss to the solid residue by premature precipitation of rare earth phosphates. It is thought that as the pH increases, iron phosphate precipitates. If there is not enough iron to react with the phosphate in the first instance, then the phosphate will combine with rare earths and precipitate as rare earth phosphate. The Nolans Bore feed to the acid bake has a molar ratio Fe:P = 0.6, and therefore some iron addition to the acid bake or water leach liquor would be desirable in order to minimise rare earth losses in the neutralisation step.

The results of a variety of neutralisation tests are set out in Table 6.

Post-neutralisation liquor samples were now treated with ammonium carbonate to precipitate a composite rare earth carbonate, a commercial end product suitable for further processing by known techniques to separate out the rare earth components.

The composition of the composite rare earth carbonate is set out in Table 7. One optional further treatment, especially for the purpose of further reducing the aluminium and thorium content, is to leach this carbonate with hydrochloric acid to produce a rare earth chloride end-product.

The earlier mentioned rare earth phosphates precipitated from the pre-leach liquor were split into samples. Some were successfully treated in a process stream similar to that outlined above for the pre-leach residue, but other processing options may be more useful in extracting the rare earths from the phosphates.

Table 1

Analysis of Nolans Bore High Grade Ore (%)

Component (%)	XRF Assay
Al	1.9
Ba	0.051
Ca	26
Ce	2.7
F	0.9
Fe	1.7
La	1.1
Nd	1.3
P	12
Pr	0.45
Si	7.8
Sm	0.16
Sr	0.45
RE (La-Sm)	5.7
Th (XRF)	0.55
Th (NAA)	0.53 (21.6 Bq/g)
U (XRF)	0.049 (6.0 Bq/g)
U (DNA)	0.034
Y	0.10

Table 2

HNO$_3$ Pre-Leach of Nolans Bore Ore – 54% w/w HNO$_3$

880 kg/t HNO$_3$, uncontrolled temperature

Leach Conditions

Initial Wet Mass Ore (g) = 282.5

Initial Dry Mass Ore (g) = 250.2

% Moisture = 11.4

Entrained water = 32.3

Added water (mL) = 56.5

HNO$_3$ volume (mL) = 225.06

HNO$_3$ concentration (g/L) = 979.8 (69% w/w)

Density of Conc. HNO$_3$ (g/mL) = 1.42 (69% w/w)

Initial wt% solids = 38

HNO$_3$ (kg/ton dry ore) = 881

HNO$_3$ concentration initially = 54.0 % w/w or -11M

Leach temp range (°C) = 22-60°C (uncontrolled)

Comments: Dispensed acid into slurry at 10 mL/min, no temp. control.

No precipitate formed in pregnant liquor after filtration

2 blade, pitched impeller, baffled, glass leach vessel.

Stirring ok. A lighter, foamy layer formed on the top of the slurry but raising the agitator resulted in better mixing.

Element	Sample wt%	Liquor Analyses (mg/L) 2 h	Solid Analysis (wt%)	%Dissolution	
			Final Residue 2 h	Liquid Final	Solid Final
Al	1.83	2880	4.83	21	24
Ca	25.1	170000	2.93	90	97
Ce	2.66	7190	5.34	35	42
F	0.9		0.60		81
Fe	1.59	2180	4.56	18	18
La	1.06	2262	2.52	28	32
Nd	1.27	3329	2.40	35	46
P	11.44	77400	2.58	89	94
Pr	0.44	976	0.76	30	50
Si	7.61	3490	24.52	6	8
Sm	0.15	487	0.28	42	48
Sr	0.44	1810	0.37	55	76
Th	0.55	1480	0.99	33	48
U (XRF)	0.05	160	0.08	43	50
Y	0.10	272	0.11	38	67
LREE (La-Sm)	5.58	14245	11.30	34	42

Table 3

HCl Bulk Pre-Leach of Nolans Bore Ore at 45°C

510 kg/t HCl (-110% stoich. for Ca), 28 wt% initial solids

Leach Conditions

Size distribution, P_{80} = 180 μm
Initial Wet Mass Ore (g) = 1196
490
Initial Dry Mass Ore (g) = 1401.5
% Moisture = 19.2
11.1
Entrained water (mL) = 284.5
Added water (mL) = 1056.5
HCl volume (mL) = 1928
HCl concentration (g/L) = 371 (32% w/w)
Density of Conc. HCl (g/mL) = 1.1594 (32% w/w)
Initial wt% solids = 28
HCl (kg/ton dry ore) = 510
HCl concentration initially = 20.0 % w/w
Leach temp range(°C) = not controlled (21-44°C)

Comments:

Concentrated HCl was added with a dosimate at a dose rate of 100mL/min

Maximum temperature reached was 44°C after addition of about 1664 mL HCl

A vacuum pump was used to pump gas at a slow rate from the head space over the slurry through a water trap for HF

Some white precipitate formed in the primary filtrate overnight

Density of P.F. was measured after filtering off white solid that precipitated

Mass of white solid that precipitated in the recovered P.F. was 1.4 g

White solid was washed with a small amount of water, dried in an oven and assayed by XRF

XRF assay of white solid: 25.4% Ca, 18.7% S, 1.2% Ba, 1.9% P: solid appears to be $CaSO_4$ (head contains 0.2% S)

Element	Sample wt%	Liquor Analyses (mg/L) 2 h (before ppt)	Solid Analysis (wt%) Final Residue 2 h	%Dissolution	
				Liquid Final	Solid Final
Al	1.83	1650	4.77	22	20
Ca	25.1	102000	2.66	97	97
Ce	2.66	3530	6.01	34	31
F	0.9		0.60		80
Fe	1.59	1320	4.49	21	14
La	1.06	1214	2.68	29	23
Nd	1.27	1806	2.69	37	35
P	11.44	47600	2.86	104	92
Pr	0.44	423	0.94	27	35
Si	7.61	2740	23.58	16	5
Sm	0.15	312	0.33	53	34
Sr	0.44	1120	0.43	73	70
Th	0.55	408	1.46	24	18
U (XRF)	0.05	59	0.10	32	35
Y	0.10	170	0.12	45	62
LREE (La-Sm)	5.58	7286	12.64	34	31

Table 4
Rare Earth Precipitation with NH_3 from Nitric Acid Leach Liquor After Partial Ca and F Removal

Precipitation conditions

Volume Feed (mL) = 100
Temperature (°C) = 80°C
Vol. of 25% NH_3 soln. added = 30
kg NH_3/kg (La-Sm) in solution = 3.31
Agitation time (min) = 120
pH in 1+13 dilution = 1.70
(1+13 = 1 mL sample in 13 mL D1 H_2O)

Volume Primary Filtrate (mL) = 86
Wash Volume (mL) = 211

Precipitate

Mass ppt dry (g) = 5.99
%Moisture = 78
Drying Temperature (°C) = 100
S.G of 25% NH_3 solution = 0.91

Comments: Precipitate was slow to filter so attempted to flocculate repulp slurry (Magnafloc 333) but no flocs formed
Green precipitate

Element	Feed mg/L	Assays		%Dissolution	
		ppt wt%	Barren liquor mg/L	Liquor	Solid
Al	12700	0.86	10700		4
Ca	68600	4.8	49800	6	4
Ce	10100	17.6	160	98	105
F		2.2			
Fe	7340	1.5	6080		12
La	3554	5.1	671	75	86
Nd	4975	7.7	642	83	93
P	148000	13.5	111000	3	5
Pr	1198	2.2	117	87	111
Si	3730	2.1	2130	26	34
Sm	792	1.01	128	79	77
Sr	1900	0.58	1360	7	18
Th	2140	3.7	236	86	105
U	300	0.02	177	23	4
Y	444	0.4	280	18	48
SUM (La-Sm)	20619	34	1719	89	98

Table 5

Rare Earth Precipitation with NH_3 from Hydrochloric Acid Leach Liquor

Precipitation conditions

Volume of Feed (mL) = 150

Temperature (°C) = 80

Vol. of 25% NH_3 soln. added (mL) = 15.01

kg NH_3/kg (La-Sm) in solution = 3.15

Agitation time = 120

pH in 1+13 dilution = 1.83

(1+13 = 1 mL sample in 12 mL D1 H_2O)

Volume Primary Filtrate (mL) = 143

Wash Volume (mL) = 305

Precipitate

Mass ppt dry (g) = 2.46

Drying Temperature (°C) = 100

S.G of 25% NH_3 solution = 0.91

Comments: High concentration of RE's in the wash indicates that the wash is redissolving some precipitated RE's ie Acid concentration in wash is too strong

% Precipitation based on liquor assumes no solution volume change

Element	Feed mg/L	Ppt wt%	Assays	%Precipitation	
			Liquor (2 h) (mg/L)	Liquor	Solid
Al	1650	0.75	956	36	7
Ca	101000	4.5	85800	7	1
Ce	3500	16.5	59	98	77
F		3.4			
Fe	1410	1.01	889	31	12
La	1214	5.2	50	96	70
Nd	1796	8.3	37	98	76
P	47800	9.9	38400	12	3
Pr	423	2.1	4	99	81
Si	1880	3.5	903	47	30
Sm	286	1.08	8	97	62
Sr	1090	0.34	914	8	5
Th	457	2.5	7	98	91
U	59	0.02	49	8	4
Y	159	0.62	26	82	64
SUM (La-Sm)	7219	33.2	158	98	75

Table 6
Neutralisation of Bake/Water Leach Liquor at Ambient Temperature
Neutralisation Additive: MgO

Test ID	Fe:P mole ratio	pH	%La-Sm Pptn	%Al Pptn	%Fe Pptn	%P Pptn	%Th Pptn
BN16	0.18	1.7	70	16	78	69	99
BN17	1.04	1.7	<3	0	92	91	96
BN21	1.10	2.5	2	4	92	98	99
BN23*	1.07	2.4-2.5	2	9	92	97	99
BN13	0.43	3	60	46	95	97	97
BN20	1.06	3	<5	30	93	98	99
BN22**	1.05	2.9-3.0	<5	20	94	98	99
BN19	0.73	3.4-3.0	<5	41	90	91	99
BN18	1.04	3.4	<5	67	99.7	99.6	99.7

*$MgCO_3$ added instead of MgO, **O_2 sparged

Table 7
Rare Earth Carbonate Composition

Test ID Wt%	C5	C6	C7	C8	C9
La-Sm	34	33	34	34	33
Al	5.9	5.7	5.9	5.8	5.7
Fe	0.8	0.7	0.7	0.8	0.8
Ca	0.9	0.8	0.5	0.8	0.9
S	0.8	1.6	1.4	0.8	1.9
Mg	0.5	0.9	0.2	0.4	1.1
Th	0.03	0.02	0.02	0.02	0.03

THE CLAIMS DEFINING THE INVENTION ARE AS FOLLOWS:

1. A process for recovering rare earth elements from a composite ore containing rare earth elements that includes a monazite group mineral and an apatite mineral, including:

pre-leaching the composite ore with an acid so as to substantially dissolve the apatite mineral into the leach liquor;

precipitating rare earth elements from the pre-leach liquor;

subjecting the residue of the pre-leaching step to an acid bake treatment, followed by a water leach, to produce a water leach liquor rich in rare earth elements;

separating impurities including thorium and iron from the water leach liquor by introducing a neutralising additive to said water leach liquor rich in rare earth elements; and

precipitating rare earth elements from the post-neutralisation liquor.

2. A process according to claim 1 wherein the acid employed for the pre-leach is nitric acid.

3. A process according to claim 2 wherein the initial concentration of the nitric acid is in the range 30 to 60wt%.

4. A process according to claim 2 wherein the initial concentration of the nitric acid is in the range 40 to 50wt%.

5. A process according to claim 1 wherein the acid employed for the pre-leach is hydrochloric acid.

6. A process according to claim 5 wherein the concentration of the hydrochloric acid is in the range 10 to 40wt%.

7. A process according to claim 5 wherein the initial concentration of the hydrochloric acid is in the range 20 to 30wt%.

8. A process according to any one of claims 1 to 7 wherein rare earth elements are precipitated from the pre-leach liquor as phosphates.

9. A process according to claim 8 wherein said precipitation of rare earth elements as phosphates from the pre-leach liquor is by addition of ammonia or hydrated lime.

10. A process according to any one of claims 1 to 9 wherein the conditions of said precipitation of the rare earth elements from the pre-leach liquor are selected to recover, with the rare earth elements and in a form separable subsequently, a major proportion of thorium present in the pre-leach liquor from the composite ore.

11. A process according to any one of claims 1 to 10 wherein the apatite is a fluorapatite, and a fluoride component of the pre-leach liquor is removed by addition of a sodium salt to cause precipitation of the fluoride component.

12. A process according to any one of claims 1 to 11 wherein said acid bake treatment is carried out in a rotary kiln.

13. A process according to any one of claims 1 to 12 wherein said acid bake treatment is effected for between 1 and 3 hours.

14. A process according to any one of claims 1 to 13 wherein said acid bake treatment is preferably a sulphuric acid bake.

15. A process according to claim 14 wherein said acid bake treatment is preferably effected at a temperature in the range 210-250°C.

16. A process according to any one of claims 1 to 15 wherein said thorium and iron separation is effected by neutralisation with a suitable alkali or alkaline earth additive selected to minimise rare earth co-precipitation or absorption.

17. A process according to claim 16 wherein said neutralising additive is a magnesium compound such as magnesium oxide or magnesium carbonate.

18. A process according to any one of claims 1 to 17 wherein a minimum Fe:P molar ratio of 1 is maintained in the water leach liquor feeding the neutralisation in order to prevent rare earth loss to the solid residue by premature precipitation of rare earth phosphates.

19. A process according to any one of claims 1 to 18 wherein said precipitation of the rare earth elements from the post-neutralisation liquor is by carbonate or double salt precipitation.

20. A process according to claim 19 wherein the rare earth carbonates or hydroxides resulting from said carbonate or double salt precipitation are leached to produce a composite rare earth chloride as an end-product.

21. A process according to any one of claims 1 to 20 wherein the pre-leach acid addition is about 110% stoichiometric with respect to calcium.