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Review

Rare-Earth Elements in Mine Tailings: Global Resources, Recovery Technologies, Environmental Challenges and Circular Economy Opportunities

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Abstract

Mine tailings represent large secondary inventories of rare-earth elements (REEs). However, their potential cannot be assessed from total concentration alone. Mining worldwide generates 9–14 Gt of tailings annually across approximately 8500 facilities, meaning that grades of only tens of mg/kg may represent substantial inventories. Their valorization depends on tonnage; the distribution of Nd, Pr, Dy, Tb, Sc, Y, and other REEs; host mineralogy; liberation; particle size; prior mass reduction; chemical selectivity; and the quality of the pregnant leach solution (PLS). Published cases include Fe-REE-Nb, carbonatite, phosphate, uranium-bearing, Pb-Zn, gold, and copper-bearing tailings, confirming that no universal flowsheet exists. The most robust routes integrate geometallurgical characterization, preconcentration, activation of refractory phases, selective leaching, PLS purification, and recovery by solvent extraction, ion exchange, membranes, adsorption, or precipitation. Major environmental challenges include Th and U mobilization, acid drainage, Fe-Al-Ca-Mg-P co-dissolution, water and reagent consumption, salt generation, and the reactivity of post-extraction residues. Circular economy opportunities increase when co-products, rehabilitation, reagent recycling, and the safe reuse of mineral residues are integrated. A decision framework is proposed that distinguishes geochemical inventory, potential resource, and recoverable inventory, prioritizing value per unit mass treated, PLS quality, and reductions in environmental liabilities. The value per tonne depends particularly on the proportions of Nd, Pr, Dy, and Tb and may range from approximately one dollar to more than a thousand. The transition to continuous circuits, comprehensive balances, and techno-economic and life cycle assessments will determine which tailings can become competitive secondary sources of REEs.

Keywords: critical raw materials; secondary resource valorization; geometallurgy; hydrometallurgy; selective leaching; techno-economic assessment

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1. Introduction

Rare-earth elements (REEs) comprise the lanthanides, together with Sc and Y, and are essential inputs for permanent magnets, advanced electronics, catalysts, alloys, optical systems, and numerous decarbonization technologies. Their criticality stems not from uniform

geological scarcity but from the combined effects of geographically concentrated extraction and refining, complex separation processes, long lead times for developing new operations, and substantial differences in the value of individual REEs. The demand associated with Nd-Fe-B magnets places particular importance on Nd, Pr, Dy, and Tb, whereas Sc and Y may determine the economic attractiveness of certain secondary materials [1–8].

Wind power deployment scenarios project an increasing and geographically concentrated demand for Nd and Pr toward 2040, and international renewable energy agencies identify magnet rare earths as one of the critical material groups most exposed to supply bottlenecks during the energy transition [9,10].

Supply security has acquired an additional dimension during 2025–2026. The International Energy Agency projects sustained growth in the demand for magnet rare earths through 2040 and warns that concentration is particularly high in refining and magnet manufacturing. Although mining capacity outside the leading producer is expanding, the development of separation, metallization, and magnet manufacturing capacity is progressing more slowly. This asymmetry makes secondary sources strategically important where they can be incorporated into regional processing circuits, provided their recovery does not entail disproportionate environmental impacts [11,12].

At the same time, mining has accumulated enormous volumes of tailings generated by comminution, concentration, and separation of valuable minerals. These deposits contain material that has already been extracted, transported, and, in many cases, finely ground, and are therefore often described as "pre-mined" resources. This advantage is real, yet only partial. Tailings are not automatically preconcentrated ore: they may contain REEs in refractory phases, ultrafine particles, complex associations with Fe oxides, silicates, phosphates, or carbonates, spatial heterogeneity, secondary oxidation, and an unfavorable ratio of mass to be treated to recoverable value [13]. Recent reviews on circular economy and critical material recovery from tailings recognize their potential while emphasizing the need to integrate processing with the ultimate residue disposition [14–22].

The scale of the inventory justifies the interest while demanding care in the calculations. Global annual tailings generation has been estimated between 8.85 and 14.4 Gt, and the most comprehensive inventory of disclosed facilities includes 1743 deposits containing approximately 44.5×10^9 m^3, within an estimated total of approximately 8500 active, inactive, and closed facilities [23,24]. At an annual generation rate of 10 Gt and total REE grades of 50–300 mg/kg, the geochemical content incorporated into new deposits each year would be 0.5–3 Mt, equivalent to one to eight times the global primary rare-earth oxide production [12]. This comparison is useful as an indication of scale but misleading as a conclusion: it describes content rather than a resource because the actual recoverable fraction is the product of four successive yields, and only a minority of facilities combine grade, mineralogy, tonnage, and accessibility in ways compatible with a viable project.

The literature specifically addressing REEs in tailings has grown rapidly. Documented cases include carbonatite tailings with REE grades of several percent, iron tailings from Bayan Obo containing monazite and bastnaesite, phosphate residues dominated by apatite, uranium tailings with a water-soluble REE fraction, gold tailings in which recovery improves after Fe removal, and copper porphyry tailings with grades on the order of tens of mg kg^{-1}. The diversity of these matrices demonstrates that a universal technological route does not exist [25–31].

The central scientific challenge is shifting the question from "What quantity of REEs do the tailings contain?" to "In which phases and particle sizes do they occur, what fraction can be concentrated before leaching, how much of that value can be selectively dissolved, what composition will the pregnant leach solution (PLS) have, and what will happen to the remaining residue?" Under this approach, recoverability is an emergent property

of the mineralogy–process system, rather than a direct consequence of grade. The same total concentration can yield radically different outcomes depending on whether REEs are adsorbed onto surfaces, hosted in apatite, encapsulated in submicron monazite, or incorporated into other phases by isomorphous substitution.

Rigorous comparisons require four concepts to be distinguished. Geochemical content corresponds to the total mass of REEs present; potential inventory combines dry tonnage and grade; a technical resource requires sufficient spatial, mineralogical, and metallurgical information; and recoverable inventory accounts for losses during preconcentration, leaching, separation, and final production. In simplified terms, the recoverable inventory can be expressed as the product of dry tonnage, grade, and four stage recoveries:

$$RI = T \times G \times R_{pre} \times R_{leach} \times R_{sep} \times R_{prod} \quad (1)$$

where RI is the recoverable inventory (t of REEs), T is the dry tonnage of tailings (t), G is the total REE grade expressed as a mass fraction, and R_{pre}, R_{leach}, R_{sep}, and R_{prod} are the fractional recoveries of preconcentration, leaching, separation, and final production, respectively. This distinction prevents transforming results from isolated samples into resource estimates and allows the comparison of deposits with markedly different tonnages, grades, and mineralogies.

Available reviews of tailings valorization and critical material recovery describe technologies and case studies in detail, but they rarely connect within a single framework the scale of inventory, the quality of evidence supporting each published claim, and the economic threshold that each tonne treated must meet [14,17–20]. This paper integrates these three dimensions: it organizes global cases by deposit family and level of evidence, translates grade and elemental distribution into value per tonne under explicit assumptions, and structures decision-making around a stage-gate sequence at which studies can be discontinued before investing in chemical optimization.

2. Global REE Resources in Mine Tailings

2.1. Tailings as Secondary Inventories and the Limits of the Resource Concept

The conceptual advantage of tailings over primary ore is that the rock has already been extracted, transported, and comminuted. Therefore, a portion of the energy and capital associated with initial mining has already been expended. However, tailings also concentrate the losses from historical processing: minerals that did not respond to separation, particles that were too fine, complex mineral associations, and elements that lacked value at the time of operation. Technological advances and market price shifts may transform these losses into opportunities, but the presence of a surface deposit does not eliminate the costs associated with rehandling, dewatering, classification, rehabilitation, permitting, or risk management [14–16,19].

There is no homogeneous global inventory of REEs contained in tailings that is comparable to primary mine production statistics. Tailings databases mainly describe location, volume, stability, or the main commodity, whereas total rare-earth element (TREE) content, total rare-earth oxide (TREO) content, and individual REE distributions are rarely reported consistently. Moreover, legacy deposits often lack three-dimensional models and sampling programs designed according to geometallurgical criteria. Consequently, a responsible global assessment must rely on archetypes and representative case studies rather than an aggregate tonnage estimate lacking quantified uncertainty [14,32].

Defining the limits of this concept also requires distinguishing actual mine tailings from other secondary industrial residues containing significant REE concentrations. Bauxite residue (red mud) from alumina production and phosphogypsum from the phosphoric acid and fertilizer industries share with tailings the characteristics of having already been

processed, occurring in large volumes, and being surface-available. However, they originate from chemical transformations subsequent to mineral beneficiation and are subject to alkalinity or acidity balances and regulatory frameworks of their respective industries [33,34]. They therefore fall outside the strict category of mine tailings, although their shared metallurgical and environmental challenges allow leaching and purification routes to be compared.

This distinction can also be expressed through a standardized classification. The United Nations Framework Classification for Resources categorizes any project along three axes: social and economic viability, project maturity, and confidence in the estimate and explicitly includes anthropogenic resources, allowing tailings deposits to be classified using the same terminology as primary deposits (United Nations Economic Commission for Europe, 2020). Regulation (EU) 2024/1252 adopted this terminology by requiring Member States to establish databases of closed extractive waste facilities and classify them according to this framework where possible [35]. Adopting a recognized classification prevents the average of a handful of samples from being presented as a resource and enables comparisons between deposits assessed by different teams with varying sampling densities and levels of metallurgical testing.

2.2. Mineralogy, Value Basket, and Global Typologies

Process mineralogy must precede lixiviant selection. Monazite, xenotime, bastnaesite, florencite, apatite, and Sc-bearing phases respond differently to metallurgical treatment. Monazite may require alkaline or sulfuric acid (H_2SO_4) activation to break its phosphate structure; bastnaesite combines carbonate and fluoride and may cause fluoride-related problems; apatite can serve simultaneously as a source of P and REEs; and florencite commonly occurs in altered, ultrafine systems. Mineralogical characterization must resolve not only major phases by XRD, but also REE distribution using SEM-EDS, EPMA, QEMSCAN/MLA, or microanalytical techniques where host phases are scarce [36–44].

Total REE concentrations must be accompanied by individual elemental distributions. Tailings enriched in Ce and La may have a different economic rationale from those with lower TREE content but higher proportions of Nd-Pr or heavy rare-earth elements (HREEs). Similarly, Sc may constitute a high-value co-product even at low concentrations. Assessments should report individual element contents, equivalent oxide contents, and potential value subject to recovery, avoiding the conversion of spot prices into "in situ value" without accounting for process losses, purity, and separation costs. The presence of Th and U requires dedicated mass balances because their mobilization during roasting, leaching, or extraction can alter the radiological classification of concentrates and residues [45–49].

The most promising mine tailings can be grouped into six families. The first comprises tailings generated by operations that already process REE minerals or REE-rich polymetallic systems, where residual grades may remain significant (Bayan Obo and certain carbonatites). The second includes tailings from other commodities in which REEs were unrecovered co-products, such as phosphates, uranium, Fe, and some base metals. The third encompasses low-grade, large-tonnage tailings, including certain porphyry coppers, where economics depend on exceptional selectivity or integration with another reprocessing objective. The fourth consists of legacy deposits where recovery is combined with environmental rehabilitation and the economic benefits must account for avoided closure and remediation costs [15,21,26,28]. The fifth gathers coal-derived residues (fine tailings, coarse refuse, fly ash, and calcination products), in which REEs are associated with clays, Fe oxides, and secondary phases generated by combustion or roasting, and where the annual scale of generation is comparable to or greater than that of many metal mine tailings [50–55].

The sixth family groups heavy mineral tailings, particularly residues from tin mining, where monazite and xenotime were discarded together with zircon and ilmenite during

gravity, magnetic, and electrostatic separation of cassiterite. These materials are atypical because their grades may reach a variety of percentages: in tailings from Bangka Island (Indonesia), REE contents of 3.7–20.3% have been measured depending on particle size fraction, with a published district range of 0.6–15.5%. Fractions above 75 µm can yield concentrates containing 40 to 68% TREO through conventional physical beneficiation [56]. This advantage has two limitations: the material's radioactivity requires radiological controls starting at the laboratory scale, and the fraction below 75 µm contains concentrated ultrafine monazite that conventional magnetic separation cannot recover.

The dominant mineralogy of each family largely reflects the primary deposit type from which the tailings originate. Carbonatite systems and alkaline igneous rocks contribute bastnaesite, monazite, and accessory minerals enriched in light rare-earth elements (LREEs); iron-phosphorus deposits and granitic pegmatites contribute apatite and xenotime; and placer and heavy mineral sand provinces add monazite, xenotime, and resistate minerals to the tailings mineralogical assemblage [57–59]. This correspondence between primary geology and residual mineralogy makes it possible to anticipate which family of recovery pathways is most likely to succeed, even before detailed characterization is complete.

2.3. Representative Case Studies Worldwide

Bayan Obo, China, is the archetype of polymetallic tailings with high strategic value. Decades of prioritizing Fe and partial REE recovery left tailings containing REEs, Nb, F, Sc, Th, and other components. Their complexity lies not only in grade but also in the intergrowth of bastnaesite and monazite with Fe minerals, fluorite, and gangue. The literature spanning 2017–2026 shows a transition from direct leaching toward mineral reconstruction, magnetic separation, flotation, and activation routes, while a recent review emphasizes the need for integrated Fe-REE-F-Nb-Sc-Th recovery, rather than optimization for a single metal [60–64].

Carbonatite systems illustrate a distinct contrast. Sarker et al. [31] characterized tailings with high REE and Fe concentrations, in which more than 70% of both the mass and the REEs resided below 63 µm, and the host minerals were closely associated with goethite. In Finland, Hajdu-Rahkama et al. [30] observed a strong dependence on particle size and significant REE losses during separation based on density and magnetic properties. These findings demonstrate that an attractive grade does not guarantee a physical concentrate, particularly when the valuable fraction is ultrafine.

Uraniferous tailings introduce an additional dimension: a fraction of the REEs may occur in secondary or soluble phases formed during processing and environmental weathering. Levett et al. [28] demonstrated water leaching recovery and estimated a potential inventory exceeding 10,000 t of REEs for the target deposit, shifting the processing rationale from destroying a refractory mineral lattice to recovering species that have already been mobilized. Concurrently, the coexistence of U and Th requires radiological balances and mobility controls starting from initial deposit characterization.

In abandoned Pb-Zn tailings in Spain, total REE concentrations are significantly lower, but combining recovery and remediation creates another source of value. Medina-Díaz et al. [22] evaluated electrokinetically assisted phytoextraction, while Muñoz-Morales et al. [21] integrated resource inventorying, life cycle assessment, and cost analysis. This case demonstrates that a process may deliver modest metallurgical performance yet remain relevant as part of restoration; it also shows that the economics of recovering low-grade REEs are challenging when assessed solely against the value of the oxides obtained.

Porphyry copper tailings represent the large-tonnage, low-grade extreme. Santana et al. [26] reported 61.74 mg/kg LREEs and 6.8 mg/kg HREEs in Chilean tailings and evaluated deep eutectic solvents, achieving a maximum overall recovery of approximately 44% with choline chloride–p-toluenesulfonic acid (ChCl-pTSA), while ChCl–lactic acid

exhibited higher relative affinity for some HREEs. This outcome is significant as it proves leachability in a real matrix, but it also illustrates the need to concentrate value or integrate REE recovery into a larger reprocessing flowsheet. Likewise, a study conducted at Arturo Prat University in Chile reported 83.16 mg/kg LREE and 12.99 mg/kg HREE in Chilean porphyry copper tailings and that froth flotation successfully concentrated light and heavy rare-earth elements contained in these tailings, outperforming gravity separation processes using JIG and a Knelson Concentrator. Results are especially attractive for some heavy rare-earth elements like Dy, Gd and Tb, because they showed an increase in concentration in an RE-enriched concentrate of 60, 52.5 and 50%, respectively.

Coal-derived residues constitute the fifth major family of secondary REE sources and, unlike metal mine tailings, originate from coal mining and preparation. In the United States, studies of coarse refuse from coal preparation plants document integrated flowsheets combining flotation, activation, and leaching, together with fractional precipitation of REE from the PLS [54,55]. Following calcination, the same type of material allowed REEs to be recovered alongside Co, Cr, Li, Mn, Sr, and V [53], while Th and U separation from a stripping solution generated from coarse refuse demonstrated that these residues require the same radiological oversight as monazite-bearing metal mine tailings [52]. In coal preparation fines, an organic lixiviant extracted a substantial amount of REE [65]. More recent studies on calcination followed by acid leaching and selective grinding combined with alkali roasting of carbonaceous gangue show additional gains when thermal pretreatment is optimized, consistent with findings for calcined coarse refuse [51,66,67]. For fly ash, sequential leaching via chemical roasting, water washing, and acid leaching, as well as deep eutectic solvents as lixiviants followed by oxalate or fluoride precipitation, offer alternative pathways with lower consumption of concentrated mineral acids [50,68]. Collectively, this family demonstrates that the generation scale of coal waste and the solids-handling infrastructure available in active coal basins may offset moderate REE grades.

Heavy mineral tailings lie at the opposite extreme from porphyry coppers. Park et al. [56] studied Bangka tin tailings containing 3.4% TREO in the fraction below 75 μm, with a dose rate on the order of 0.1 μSv/h, demonstrating that the primary constraint is particle size rather than grade: monazite with a D50 of 3 μm failed to respond to conventional magnetic separation. This case shows that, for this family, the processing challenge shifts from selective dissolution to the physical capture of ultrafines and thorium management, and that a single deposit may be both the richest and the most constrained by regulation.

Table 1 summarizes these representative case studies, indicating for each one the resource characteristics, the mineralogy or condition of the tailings, and the main implications for REE recovery. Taken together, the cases show that the governing constraint changes from one family to another: ultrafine particle size in carbonatite and tin tailings, radiological control in uranium-bearing and heavy mineral tailings, and high selectivity or integration with reprocessing and remediation in low-grade porphyry copper and Pb-Zn tailings.

Table 1. Representative global case studies of REEs in mine tailings and their primary processing implications.

Region/Tailings Type	Resource Characteristics	Mineralogy/Condition	Recovery Implications	Reference
China, Bayan Obo (Fe-REE-Nb)	Polymetallic tailings containing REEs, Nb, F, Sc, and Th; significant residual grades in specific streams	Bastnaesite + monazite closely associated with Fe and fluorite	Favors preconcentration, mineral transformation, and multi-product recovery prior to intensive leaching	[60,62,64]
Australia, carbonatite tailings	Relatively high-grade case study, with mass and REEs strongly concentrated in fines	Monazite/florencite associated with goethite	Ultrafine mineralogy limits physical separation: hydrometallurgy may be required	[31]

Table 1. *Cont.*

Region/Tailings Type	Resource Characteristics	Mineralogy/Condition	Recovery Implications	Reference
Finland, Pb-REE carbonatite tailings	Nd-Pr and other REEs with size-dependent distributions	REE minerals mixed with silicate gangue and oxides	Multistage separation: substantial losses if only density or low-intensity magnetic separation is used	[30]
Australia, uranium tailings	Water-soluble REE fraction; significant potential inventory	REEs in secondary/soluble phases developed during processing and environmental weathering	Potential for low-intensity recovery; U-Th balance mandatory	[28]
Spain, abandoned Pb-Zn tailings	TREE values on the order of hundreds of mg/kg and strong rehabilitation potential	REEs dispersed in an altered polymetallic matrix	Phytoextraction and electrokinetic methods are slow but can be integrated with remediation	[21,22]
Chile, porphyry copper tailings	Tens of mg/kg; LREEs > HREEs in the published case studies	REEs dispersed within a refractory silicate matrix	Requires high selectivity, low consumption, and preferential integration with reprocessing for other metals	[26]
Algeria, fine phosphate mining residues	~0.64 wt% LREE in a target phosphate stream	Fluorapatite and carbonate-fluorapatite	REE recovery must be integrated with Ca-P-F and phosphogypsum management	[69]
REO-rich iron-oxide-silicate tailings	REEs associated with hematite, quartz, and phosphate minerals	Monazite and REE phases within Fe-Si gangue	Hydroxamates and selective flotation are feasible where liberation is sufficient	[36–38]
United States, coal-derived residues (coarse refuse, fines, and fly ash)	Large-scale generation from coal preparation and combustion; moderate REE grades with associated Th/U	REEs in clays, Fe oxides, and secondary calcination or combustion phases	Thermal activation and acid leaching with fractional precipitation; Th/U radiological oversight mandatory	[52–55]
Indonesia, tin tailings (heavy minerals)	3.4% TREO below 75 µm; 3.7–20.3% REE depending on particle size fraction	Ultrafine monazite and xenotime alongside zircon and ilmenite	High grade with size-constrained recovery; carrier-assisted separation and radiological oversight mandatory	[56]

2.4. Heterogeneity, Aging, and Resource Estimation

Tailings deposits are not homogeneous bodies. Historical changes in ore, mineralogy, reagents, grind size, and primary recovery create chemical and particle size stratification. Sulfide oxidation, Fe/Mn redistribution, carbonation, salt precipitation, and weathering modify REE speciation and mobility over time. Consequently, inventories must be established through three-dimensional sampling programs and geometallurgical domains rather than isolated composite samples. Studies on ion-adsorption clay tailings reveal REE redistribution and Ce oxidation states associated with fine matrices and secondary oxides, reinforcing the need to incorporate post-depositional processes [70–73].

Sampling must be designed as a resource assessment program rather than an isolated metallurgical sampling exercise. Domains must capture changes in feed ore, decades of operation, particle size, reagent types, discharge zones, hydraulic segregation, oxidation, and saturation. In historical deposits, uncertainty in volume may be comparable to uncertainty in grade. Photogrammetry, near-surface geophysics, sonic or low-disturbance drilling, depth-resolved analyses, and geostatistical models enable domains to be constructed and

subsequently linked to metallurgical responses. Without this step, an average TREE value may conceal zones with markedly different economics [21,74].

2.5. Global Scale and Content Calculations

The discussion of tailings as an REE source becomes more precise when grade is translated into the mass that must be handled. Recovering one tonne of REEs requires moving, treating, washing, and disposing of a quantity of solids inversely proportional to grade and overall recovery. This quantity, rather than concentration, determines reagent, water, and energy consumption; thickener and filter sizes; and final residue volume.

Table 2 applies this calculation to the reference grades of the described families, assuming an overall recovery of 40%, which corresponds to a realistic product of four stages (preconcentration, leaching, separation, and final production). The resulting range spans almost three orders of magnitude, from tens to tens of thousands of tonnes of tailings per tonne of REE recovered.

Table 2. Mass of tailings requiring treatment per tonne of REE recovered, according to the reference grade of each family.

Reference Family	Reference Grade (mg/kg REEs)	REE Content per Mt of Tailings (t)	Tailings to Treat per Tonne of REE Recovered, R = 40% (t)	Grade Reference
Porphyry copper	70	70	36,000	[26]
Abandoned Pb-Zn tailings	300	300	8300	[22]
Coal-derived residues and low-grade tailings	1000	1000	2500	[54,55]
Phosphate residues and carbonatites	6400	6400	390	[69]
Tin tailings (heavy mineral sands)	34,000	34,000	74	[56]

Source: Authors' own elaboration. The overall recovery of 40% is the product of four stage-specific recoveries; different values scale the final column inversely.

Two immediate consequences follow. First, no route treating the entire bulk mass can be sustained at the low-grade extreme: at 70 mg/kg, each tonne of product requires processing approximately 36,000 t of tailings, multiplying any specific acid, water, or energy consumption by that factor. Second, preconcentration does not compete with leaching, but it dictates its viability, as rejecting 70% of the mass while retaining 80% of the value reduces the mass flow entering the chemical stage by more than a factor of three. In higher-grade families, this relationship reverses, and the bottleneck shifts to selectivity, Th and U oversight, and product specifications.

The chain linking geochemical content to recoverable inventory admits a direct graphical representation. Figure 1 shows how, for a feed of 1 Mt at a reference grade, the contained tonnage decreases successively during preconcentration, leaching, separation, and final product recovery, and how each stage diverts mass into residue streams that must also be characterized. The stage yields adopted in Figure 1 (R_{pre} = 0.80, R_{leach} = 0.75, R_{sep} = 0.85, and R_{prod} = 0.80) are illustrative values within the ranges reported for real tailings and tailings-derived solutions. Carrier-assisted magnetic preconcentration of tin tailings recovered 80% of TREO [56], whereas density-based and magnetic separation of carbonatite tailings incurred substantial size-dependent REE losses [30]. Leaching yields on authentic tailings range from approximately 44% with deep eutectic solvents to approximately 95% after alkaline pretreatment, the latter accompanied by the dissolution of most

of the solid mass [26,31]. Downstream, Fe co-extraction penalizes solvent extraction from tailings leachates [25], and staged precipitation from a coal refuse leachate recovered more than 80% of the dissolved REEs [54]. The resulting overall recovery of approximately 40% should therefore be regarded as a mid-range scenario rather than an optimistic one.

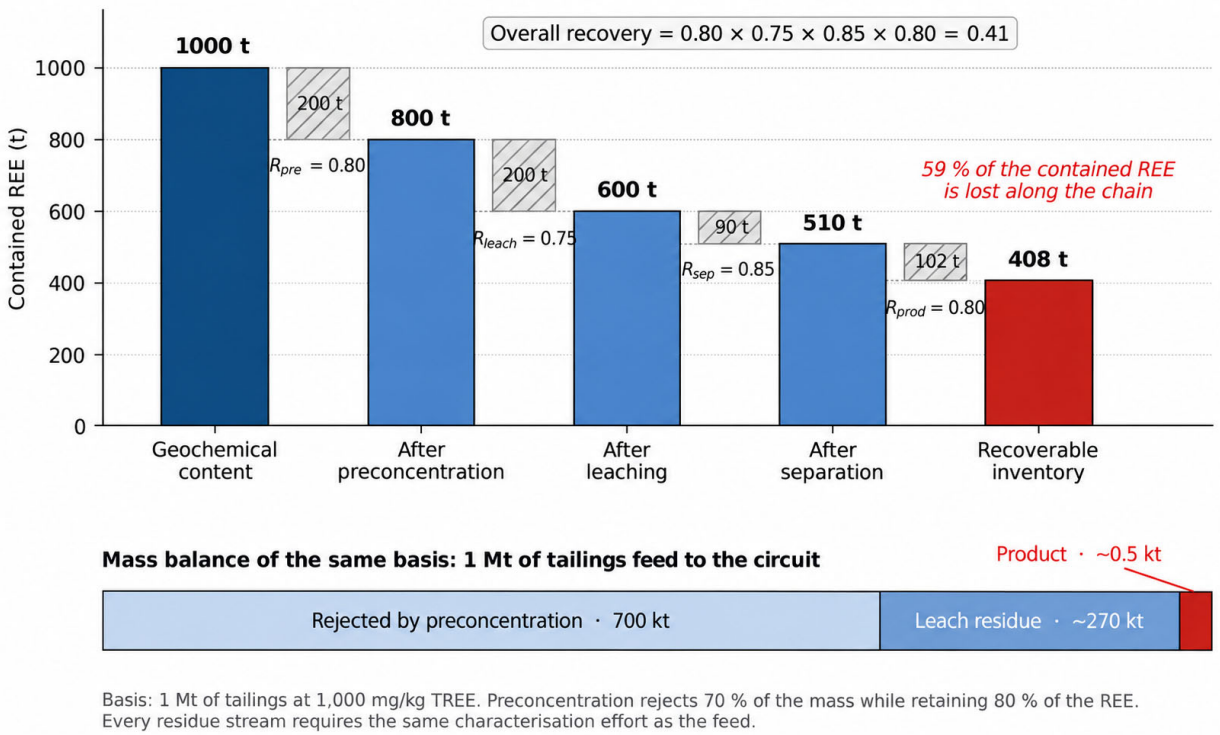


Figure 1. Inventory waterfall diagram: From geochemical content to recoverable inventory, showing losses at each stage and the associated residue streams. Stage yields are illustrative values consistent with published results [25,26,30,31,54,56].

2.6. Levels of Evidence for Published Claims

Claims regarding REEs in tailings are not directly comparable because they rest on markedly different amounts of evidence. Content measured in three surface samples and recovery measured with a closed-balance circuit occupy the same space in a summary table; yet they support decisions at substantially different magnitudes. A scale of evidence levels makes this difference explicit without dismissing exploratory studies, which remain the legitimate starting point for any resource evaluation.

Table 3 defines five ascending levels, ranging from the geochemistry of isolated samples to verified continuous operation, and assigns representative case studies from recent literature to each. The scale describes what is known, while the gate framework defines what to decide on that basis, making the two complementary.

The distribution of the literature across this scale explains the gap between expectations and industrial reality. Most research falls within E2 and E3, few studies reach E4, and virtually none attain E5 outside the multi-product circuits of Bayan Obo, where REE recovery relies on an existing iron operation. This concentration does not invalidate the findings: it indicates that decisions are being made on intermediate-level evidence, and that the highest research return lies in advancing a select number of well-characterized cases from E3 to E4 and E5 rather than adding new materials at E1.

Table 3. Levels of evidence for published claims about REEs in mine tailings.

Level	What Is Demonstrated	Minimum Data Required	Representative Cases and References
E1. Geochemical content	That the material contains REEs	Individual REE in point samples, without tonnage or mineralogy	Tailings inventories and exploratory sampling programs [15]
E2. Process characterization	Where value occurs and why it is difficult to extract	Quantitative mineralogy, size-fraction deportation, and mineral associations	[29–31]
E3. Batch metallurgical response	That a fraction of the value can be mobilized	Element-specific recovery, test conditions, and reagent consumption	[26,28,75]
E4. Bench-scale integrated circuit	That a flowsheet exists with defined products and residues	Closed mass balance, PLS quality, product purity, and residue characterization	[55,56,76]
E5. Verified continuous operation	That the system is reproducible, evaluated, and capable of planned closure	Continuous piloting campaigns, separation cycles with real matrices, techno-economic and life cycle assessments, and final residue disposal plans	No published cases for REEs from tailings outside the multi-product circuits of Bayan Obo [60]

Source: Authors' own elaboration.

2.7. Position Relative to Other Secondary Sources of REEs

Tailings compete for attention, capital allocation, and separation capacity with other secondary streams targeting the same elements. Comparisons are frequently framed in terms of grade; however, this variable alone invariably favors end-of-life magnets and conceals the differences that ultimately determine project viability. A useful comparison requires at least five dimensions: grade, annual generation volume, the proportion of value contained in magnet elements, the demonstrated maturity of the route, and the dominant scale-up obstacle in each case.

Table 4 synthesizes these dimensions for the five streams most extensively covered in the literature: end-of-life magnets, bauxite residue, phosphogypsum, coal ash, and mine tailings. The figures indicate orders of magnitude rather than project-specific metrics, as each family exhibits substantial internal variability.

Taken together, these data show that no stream dominates across all five dimensions, and that the options are not mutually exclusive. End-of-life magnets offer the highest grades and most favorable compositions, since their REE content consists almost entirely of magnet elements. However, their volume is constrained by collection and the one-to-two-decade lag between equipment sale and retirement, so they cannot meet demand growth alone. Bauxite residue, phosphogypsum, and coal ash provide volume and concentrations comparable to those of many mine tailings, yet they share two limitations: their value lies in Sc, Y, and LREEs rather than magnet elements, and each faces a distinct barrier that is difficult to circumvent, whether acid consumption and dissolved iron, radiological regulation due to radium, or encapsulation in aluminosilicate glass.

Mine tailings occupy a distinct position due to three reasons. First, their grade range spans four orders of magnitude, encompassing deposits as lean as porphyry copper tailings and as rich as heavy mineral sand tailings. Second, their mineralogy may include monazite, xenotime, and bastnaesite carriers enriched in Nd, Pr, Dy, and Tb, precisely the fraction that other high-volume streams do not provide. Third, their environmental balance differs: recovery from other sources adds a process to a stable or already managed waste stream,

whereas recovery from tailings may reduce an existing liability, allowing avoided closure costs to be included in the assessment. These advantages do not mean that tailings can be valorized without further processing. Tailings were discarded precisely because their REE content was uneconomic or could not be recovered with the original flowsheet, so any recovery requires additional or different treatment (regrinding, preconcentration, activation, or selective leaching) that incurs costs, consumes reagents, water, and energy, and generates new residues. The credit for avoided closure costs is valid only where reprocessing demonstrably reduces the liability of the deposit rather than transferring it to a new residue stream, and the position of tailings relative to other secondary sources must be assessed on this net basis.

Table 4. Comparison of secondary REE sources: Grade, volume, fraction of magnet elements, readiness, and governing bottleneck.

Secondary Source	Typical REE Grade	Annual Generation	Fraction of Magnet REE (Nd, Pr, Dy, Tb)	Technological Readiness	Governing Bottleneck	References
End-of-life NdFeB magnets and manufacturing swarf	25–32 wt% REE	Small flow, from a few to tens of kt, dominated by manufacturing swarf; historical recycling rate below 1%	Virtually all; Dy and Tb present in high-coercivity grades	Highest among secondary sources: direct hydrogen decrepitation and remelting routes proven at pilot and early industrial scales	Collection, dismantling, and traceability; the lag between sale and end-of-life limits available volume	[77]
Bauxite residue (red mud)	500–1700 mg/kg total REE; 50–130 mg/kg Sc	120–150 Mt; approximately 4 Gt accumulated	Low; LREE-dominated pattern with value concentrated in Sc	Pilot and demonstration, with several European projects	Acid consumption during neutralization, extensive Fe co-dissolution, and effluent management	[33,34,78]
Phosphogypsum	200–5000 mg/kg total REE, with an MREE-enriched pattern	200–300 Mt; approximately 10^9 t cumulative legacy stacks	Moderate; significant Nd and Pr, limited Dy and Tb	Bench and pilot scale	Radiological classification: ^{226}Ra concentrates in phosphogypsum at activities of 200–3000 Bq/kg; low grade across large bulk volume	[79]
Coal ash	250–800 mg/kg, with cases above 1000 mg/kg	Several hundred Mt, with China exceeding 500 Mt	Moderate, with high proportions of Y and heavy REEs in some basins	Bench and pilot scale, with national scale-up programs	REEs encapsulated in aluminosilicate glass require alkaline sintering or concentrated acids; yield depends on Ca content	[50,51,55,68]
Mine tailings	50 to more than 30,000 mg/kg, depending on family	8.85–14.4 Gt across approximately 8500 facilities	Highly variable: absent in porphyry copper tailings, high in carbonatite and heavy mineral sand tailings	Bench scale, with few integrated circuits and no verified continuous operations	Heterogeneity and lack of comparable inventories; host mineralogy changes the processing route even between neighboring deposits	[23,24]

The comparison also has a downstream implication that is seldom discussed. The intermediate product of all these routes is a mixed REE carbonate or oxalate competing for the same separation capacity. For a separation facility, the relevant question is therefore not the original waste source, but which impurity profile it carries: Fe and Al from red mud, Ra and sulfate from phosphogypsum, Si and Al from coal ash, and Th, U, and phosphates from mine tailings. This profile, rather than origin, determines the number of purification stages and the cost per kilogram of separated oxide.

3. Process Mineralogy, Characterization, and Preconcentration

Figure 2 outlines the integrated conceptual route for REE valorization from mine tailings, from geometallurgical characterization and physical preconcentration to activation, leaching, PLS purification, and product recovery, with REE mass balances at key stages and with residue management and circular economy opportunities considered throughout the flowsheet rather than only at its end. Process mineralogy and mass reduction form the first part of this route and determine the feasibility of all subsequent chemical stages.

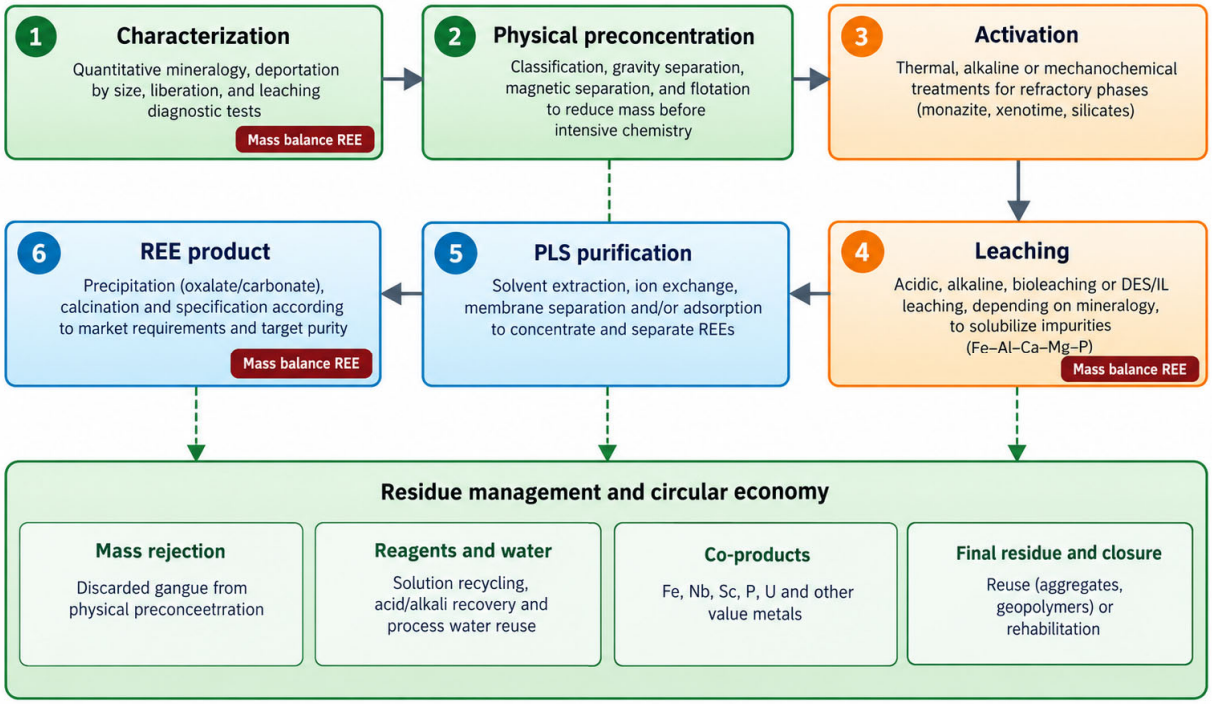


Figure 2. Integrated conceptual route for REE valorization from mine tailings, from characterization to product, with residue management and the circular economy integrated throughout.

3.1. Host Minerals and Accessibility

Mineralogical characterization must lead to a processing hypothesis. The decisive question is not only which minerals contain REEs, but rather what fraction of each element is exposed, liberated, or encapsulated; which association governs the leaching response; and which impurities will accompany dissolution. The same TREE may correspond to a relatively accessible material dominated by coatings or secondary phases, or to a refractory inventory hosted in monazite, xenotime, allanite, or to other phosphates or silicates. Mineral and size-specific deportment, combined with diagnostic leaching assays, therefore provides the link between descriptive mineralogy and flowsheet design [29,80,81].

Speciation is as important as phase identification. REEs adsorbed onto or associated with Fe-Mn coatings may respond to mild conditions, whereas structurally bound REEs in phosphates and silicates require lattice breakdown. In weathered tailings, secondary precipitation can redistribute REEs from primary host minerals onto surfaces or amorphous phases. Techniques such as MLA/QEMSCAN, SEM-EDS, micro-XRF, LA-ICP-MS, and absorption spectroscopy can establish where the value occurs and which impurities will accompany it into the PLS [29,31].

3.2. Liberation, Particle Size, and Ultrafine Particles

Particle size exerts a dual influence. Historical grinding increases specific surface area and may improve kinetics; however, slime generation reduces flotation selectivity

and increases hydraulic entrainment, reagent consumption, and thickening operations. Recent studies on ultrafine tailings demonstrate REE enrichment in fractions below 10 μm, governed by phosphates such as florencite/monazite, precisely where conventional physical concentration technologies are least efficient [29,36,38,40,42–44]. Consequently, a grade–recovery curve for each particle size fraction is more informative than a single bulk grade.

Studies published in 2026 reinforce that the ultrafine challenge is not merely operational. Offei Adu et al. [29] demonstrated REE enrichment in ultrafine fractions of flotation tailings and linked low beneficiation efficiency to ultrafine phosphate hosts. Hajdu-Rahkama et al. [30] observed a similar response in Finnish carbonatite tailings. Under these conditions, additional grinding may deteriorate thickening, filtration, and entrainment losses; the relevant question becomes whether to accept a fine-grained chemical feed or develop highly precise selective classification.

3.3. Minimum Geometallurgical Characterization Program

A robust program must combine the following: (i) spatial and depth-resolved sampling; (ii) ICP-MS/ICP-OES analysis of individual REEs, Th, and U; (iii) quantitative mineralogy; (iv) size-by-size deportment; (v) liberation and association analysis; (vi) diagnostic leaching tests; and (vii) physical properties relevant to separation. XRD alone is often insufficient when REE hosts are minor constituents. SEM-EDS, EPMA, MLA/QEMSCAN, μ-XRF, and synchrotron-based methods can identify REE hosts in fine particles, oxidation states, and associations with Fe/Mn, phosphates, or organic matter [36–39,44,70,72,73].

Chemical characterization must also include the elements that govern reagent consumption and purification: Fe, Al, Ca, Mg, P, F, Si, Mn, and S. Tailings containing 500 mg/kg TREE with low acid demand may be more attractive than tailings containing 3000 mg/kg TREE hosted in a matrix that consumes hundreds of kilograms of acid per tonne. Therefore, initial tests should report metrics more closely aligned with industrial requirements, such as “dissolved REE per kilogram of reagent” and “REE in PLS per kilogram of total dissolved mass”.

To compare sampling programs, the analytical dataset should report TREE, TREO, and individual concentrations, together with Fe, Al, Ca, Mg, Mn, P, F, S, As, Th, and U. Metallurgical performance must be linked to particle size fractions and deposit domains rather than only to a single composite sample. Characterization must include acid generation potential, neutralization capacity, and residue leachability tests prior to and following reprocessing, so that REE recovery and liability reduction can be assessed on a unified basis.

3.4. Physical Separation and Mass Reduction

Preconcentration is advantageous when REEs are partitioned into a size or density fraction and liberation is sufficient. Classification/desliming can separate an enriched stream, though they may also lose REEs when hosts are predominantly ultrafine. Magnetic separation is useful when the matrix contains magnetite/hematite or when host phases exhibit a paramagnetic susceptibility; gravity and electrostatic separation are more effective for heavy minerals at relatively coarse sizes. Monazite and bastnaesite flotation has evolved from oleates toward hydroxamates and more selective collectors, although Ca-bearing minerals and slimes remain critical interferences [41–43,82]. Sequential combinations of physical methods (gravity, magnetic, and electrostatic separation) have also concentrated REEs from iron mine tailings without chemical reagents, confirming that physical separation circuit configuration can be as important as the individual techniques [83].

Collector selection must reflect actual surface chemistry rather than that of pure minerals. Studies on bastnaesite with hydroxamates, separation from calcite and barite, and monazite flotation from kaolinite show that pH, dissolved ions, and gangue mineralogy alter

adsorption and selectivity [36–40,42,43]. In tailings, surfaces have also been conditioned by historical reagents and oxidation, so microflotation tests must be validated on authentic bulk samples. The aim of preconcentration is not necessarily to maximize recovery; accepting a moderate REE yield loss may be optimal if the mass entering roasting or leaching is substantially reduced.

The most useful preconcentration metric is not REE recovery alone, but the ratio of value retained to rejected mass. A circuit retaining 80% of REEs while rejecting 70% of the bulk mass may be more favorable than one recovering 95% while rejecting only 10%, particularly if the subsequent stage involves acid baking or leaching with concentrated acid. Preconcentration can also remove acid-consuming minerals, magnetic Fe, or carbonate phases before they enter the hydrometallurgical circuit.

Table 5 summarizes the main REE host minerals and modes of occurrence in tailings, together with their dominant REEs, primary processing constraint, and most favorable pretreatment or route. The host mineral, rather than the tailings family, largely determines whether physical preconcentration, thermal or alkaline activation, or mild lixiviants should be prioritized.

Table 5. Common REE host minerals and their implications for flowsheet design.

Mineral/Mode of Occurrence	Dominant REEs	Primary Constraint	Favorable Pretreatment or Route
Monazite, (REE, Th)PO_4	LREE, Th	Refractory phosphate; possible radioactivity	Alkaline or H_2SO_4 roasting, acid baking, prior separation where liberation permits
Bastnaesite, $REECO_3F$	LREE	F and carbonate; roasting-dependent reactivity	Selective flotation; roasting/activation followed by leaching
Xenotime, YPO_4	Y + HREE	Highly refractory phosphate	Intensive alkaline/acid baking or highly selective routes
Florencite and secondary phosphates	Variable LREEs	Ultrafine size and association with Fe oxides	Size-resolved deportment; possible direct leaching of fines
Apatite/fluorapatite	LREE + Y	Ca-P-F dominate PLS and reagent consumption	Integrate P recovery; control Ca and F downstream
Allanite and REE silicates	LREE	Silicate lattice and high reagent consumption	Thermal/mechanochemical activation; subsequent selective leaching
REEs adsorbed onto clays or coatings	Y, HREEs, or a mixture	Desorption, readsorption, and low concentration	Mild lixiviants/ion exchange; fines control
Sc incorporated by substitution in oxides/silicates	Sc	Low concentration and structural association	Recovery as a co-product after Fe/Al removal; selective solvent extraction or ion exchange

4. REE Recovery Technologies for Mine Tailings

4.1. Physical Beneficiation as the First Stage

Physical technologies (classification, gravity, magnetic and electrostatic separation, and flotation) impose the lowest chemical burden and are therefore preferable as the first

decision gate. Their success hinges on liberation and contrasts in physical properties. At Bayan Obo, Wu et al. [62] combined preconcentration and flotation to increase REO grade substantially, whereas carbonatite cases demonstrate declining efficiency when ultrafine particles predominate. Flotation using hydroxamates and carboxylic collectors remains central for bastnaesite and monazite, although actual tailings surfaces may be altered by oxidation and historical reagents [36,37,42,43,80].

Carrier-assisted magnetic separation shows how far physical beneficiation can be extended for ultrafine feeds. By rendering monazite and magnetite hydrophobic with sodium oleate and employing emulsified kerosene as a bridging liquid, monazite particles with a D50 of 3 μm unresponsive to magnetic separation attached to 106–180 μm carriers. A rougher–scavenger–cleaner circuit recovered 80% of TREO and increased the grade from 3.4% to 9.0%, reaching 23.2% after carrier removal [56]. This result directly addresses the governing bottleneck in carbonatite and flotation tailings, although carrier recycling and performance in real slurries containing clays and residual reagents require further verification.

4.2. Thermal, Reductive, Alkaline, and Mechanochemical Activation

Alkaline roasting with NaOH/KOH breaks down phosphates and silicates and may be particularly effective for monazite, xenotime, and clay-rich materials. Flowsheets at Bayan Obo also show that reduction and mineral reconstruction can segregate Fe prior to REE recovery [61,63,64,84–86]. Mechanochemical activation introduces defects and amorphization, reducing the required temperature or lixiviant concentration [87–90]. Its viability must be weighed against grinding energy costs, particularly when the tailings are already ultrafine. Microwave or ultrasound intensification may accelerate transport and remove passivating layers, but energy efficiency must be demonstrated under continuous operation [48,88,91].

Bayan Obo provides the broadest evidence for activation. Zheng et al. [64] demonstrated bastnaesite and monazite decomposition during roasting with coal, $Ca(OH)_2$, and NaOH; Zhou et al. [84] integrated magnetizing roasting, magnetic separation, $(NH_4)_2SO_4$ activation, and water leaching; and Zhang et al. [63] developed a NaCl-$Ca(OH)_2$-coal route targeting Sc, REEs, and Nb. In these flowsheets, thermal energy is justified only when it induces a transformation that substantially reduces chemical severity or enables multi-product recovery.

Mechanochemical activation is attractive for tailings because grinding can simultaneously perform physical and chemical activation. Defect generation, amorphization, and increased specific surface area may accelerate leaching in refractory phases. Nevertheless, energy costs per tonne and fines behavior must be benchmarked against thermal routes. Intensive grinding is not automatically "green" if metallurgical gains are marginal or filtration burdens and water losses increase [19,92].

4.3. Acid Leaching and Acid Baking

Hydrochloric acid (HCl), H_2SO_4, or nitric acid (HNO_3) remain the benchmark reagents due to their widespread availability and kinetics; however, selection must account for host minerals, matrix anion balance, and downstream processing. HCl may favor dissolution of activated phosphates and simplify specific extraction steps; H_2SO_4 offers low cost and compatibility with acid baking, although it may generate double sulfates or high ionic strength; HNO_3 facilitates flowsheets where Ca is removed as nitrate, but cost and recovery constraints are decisive. The fundamentals of monazite, bastnaesite, apatite, and allanite leaching are well established [87,92–97].

H_2SO_4 baking converts refractory minerals into more soluble sulfates and may be more efficient than direct agitation leaching. Its main risk lies in replacing a liberation

problem with one of acid consumption, energy demand, and neutralization. In coal tailings, acid baking followed by water leaching improved extraction relative to acid leaching, with H_2SO_4 dosage as the governing variable [98]. In REE concentrates, apatite modifies baking behavior and may increase acid demand, while Fe phases may act as reaction sinks or modifiers [92,99,100].

Selectivity must be quantified as REE mass transferred per unit of acid and per unit of total dissolved mass. High extraction accompanied by extensive Fe, Al, or Ca dissolution may yield an REE-dilute PLS that is costly to purify. This is decisive for low-grade mine tailings: the cost of gangue acid consumption may substantially exceed the value of recovered metals. Design must integrate solution recycling, acid recovery where feasible, selective neutralization, and valorization of secondary sulfates, chlorides, or phosphates.

4.4. Alkaline Activation and Routes for Refractory Phosphates

NaOH, KOH, and alkaline salts break down the phosphate structures of monazite and xenotime and convert P into separable species. Their advantage is high mineralogical conversion; their disadvantages include alkaline consumption, salt washing, and management of streams containing P, F, and Na/K. This route is particularly relevant when value is concentrated within a mineralogical fraction that can be preconcentrated before activation. For bulk low-grade tailings, reagent costs may be prohibitive [45,96].

Combining alkaline roasting with acid leaching has performed efficiently even in unconventional matrices, such as REE-rich black sands, reinforcing the broader applicability of this route beyond pure monazite and xenotime [60].

4.5. Biohydrometallurgy and Biomining

Biohydrometallurgy may exploit microbial production of organic/inorganic acids, Fe/S oxidation, complexation, and phosphate solubilization. Studies using Acidithiobacillus, Gluconobacter, Aspergillus, and phosphate-solubilizing bacteria demonstrate extraction from monazite, uranium tailings, coal, and gold tailings [49,75,95,101–111]. Its principal strength is the use of less severe chemical conditions. Limitations include kinetics, solids tolerance, metal inhibition, nutrient management, and the large solution volumes required at low grades.

Integrating pretreatment with bioleaching is more promising than bioleaching alone. In gold tailings, oxalic acid pretreatment removed Fe and increased bioaccessibility, improving Pr, Ce, and Eu recovery [75]. In uranium tailings, S- and Fe-oxidizing consortia solubilized REEs alongside U/Th, requiring a subsequent ion exchange stage [49]. Future research must shift from batch flasks to continuous reactors or heaps, incorporate solution recycling, and express productivity as the REE mass recovered per reactor volume and time rather than only as a final percentage.

Biomining has evolved from nonspecific organic acid production tests toward metabolic control and consortium selection. Li et al. [112] compared single- and two-stage acid bioleaching on REE-rich tailings, proving that functional consortia can mobilize bastnaesite minerals. Schmitz et al. [103] demonstrated that engineering of Gluconobacter oxydans enhances biolixiviant secretion and recovery. These routes are particularly interesting for low-grade streams, but volumetric productivity, relative sterility, carbon recycling, and scalability must be demonstrated in continuous reactors [113].

4.6. Deep Eutectic Solvents, Ionic Liquids, and Solvometallurgy

Deep eutectic solvents (DESs) and ionic liquids (ILs) are attractive because their acidity, coordination, and selectivity can be tailored. However, “green” must not be assumed as an intrinsic property: component toxicity, viscosity, thermal demands, wash water, solvent

losses, and regeneration must be quantified. In Chilean copper tailings, ChCl-pTSA achieved approximately 44% average recovery, while ChCl–lactic acid exhibited higher affinity for specific HREEs [26]. In Fe-rich tailings pretreated with KOH, ChCl-pTSA achieved TREE recoveries near 95% but with ~94% bulk mass dissolution. This outcome illustrates how very high recovery may represent poor metallurgical performance when selectivity is low [31]. Advances in functionalized ILs, DESs, and extraction systems provide powerful tools but require solvent mass balances and recycling trials [94,114–121].

DESs and ionic liquids allow fine-tuning of acidity, complexation, and polarity, potentially improving selectivity relative to conventional matrices. The study of Chilean copper tailings demonstrates that these solvents are already being evaluated on authentic, very low-grade feeds [26]. Nevertheless, environmental comparisons must encompass solvent synthesis, viscosity, thermal requirements, entrainment losses, toxicity, bleed treatment, and the actual number of cycles. An "alternative" solvent represents an advancement only if it reduces impact per kilogram of product recovered. Scalability imposes a further constraint relative to conventional mineral acids. DESs are considerably more viscous than dilute aqueous H_2SO_4 or HCl, which slows mass transfer, limits the solid-to-liquid ratio, and hinders solid–liquid separation and washing of fine tailings [121]. In addition, whereas mineral acids are low-cost bulk commodities consumed in a single pass, DESs and ILs must be synthesized, recovered, and recycled with very low losses to remain competitive, a requirement that has so far prevented their industrial adoption in extractive metallurgy [122]. For high-tonnage, low-grade tailings, these solvents are therefore more realistic for treating preconcentrates or small, high-value streams than for processing the bulk mass.

4.7. Intensification, Sequential Leaching, and Integration of Operations

Sequential strategies may increase PLS concentration before separation. Kang et al. [76] developed a flowsheet combining sequential leaching with solvent extraction (SX) for low-concentration tailings, doubling the aqueous REE concentration relative to single-stage leaching and yielding high-purity LREE and HREE fractions after extraction and precipitation. This finding has broader relevance: in low-grade tailings, circuit configuration may be more decisive than maximizing extraction in a single stage.

Table 6 compares the main technologies for REE recovery from mine tailings in terms of principal advantage, critical limitation, most suitable feed, and the indicators that should be reported to allow comparisons between studies. No technology is superior under all criteria; each is suited to a specific combination of mineralogy, grade, and impurity load, which is why physical, thermal, and hydrometallurgical stages are usually combined in sequence.

Table 6. Comparison of technologies for REE recovery from mine tailings.

Technology	Principal Advantage	Critical Limitation	Most Suitable Feed	Indicators to Be Reported
Classification/gravity/magnetic separation	Low chemical burden and early mass reduction	Low efficiency for ultrafines or phases lacking property contrasts	Tailings with size-dependent partitioning or liberated minerals	Enrichment, recovery, rejected mass, energy
Flotation	Can produce REE concentrates before hydrometallurgy	Sensitivity to fines, gangue, and historical surface chemistry	Relatively liberated monazite/bastnaesite	REO grade, recovery, collector consumption, selectivity against Fe/Ca

Table 6. *Cont.*

Technology	Principal Advantage	Critical Limitation	Most Suitable Feed	Indicators to Be Reported
Roasting/thermal activation	Breaks down refractory phases and may recover co-products	Energy, emissions, and salt formation	Higher-grade tailings or preconcentrates	kWh/t, mineral conversion, recovery, treated mass
Acid baking	High conversion of refractory phosphates/silicates	H_2SO_4 consumption, energy, and neutralization	Concentrates or enriched fractions	kg acid/t, TREE in PLS, co-dissolved Fe/Al/Ca
Direct acid leaching	Simplicity and rapid kinetics	Reagent consumption by gangue and impure PLS	Accessible phases or moderate-grade tailings	REEs/kg acid, PLS concentration, selectivity
Alkaline activation	Effective for monazite/xenotime	NaOH/KOH consumption and salt washing	Phosphate concentrates	kg alkali/t, mobilized P/F, alkali recycling
Bioleaching/biomining	Mild conditions and potentially low impact	Kinetics and volumetric productivity	Low-grade tailings and bio-reactive matrices	mg REE L^{-1} d^{-1}, nutrients, recovery, stability
DESs/ionic liquids	Tunable chemistry and potential selectivity	Viscosity, cost, toxicity, and regeneration	Matrices where selectivity offsets cost	Recovery, dissolved mass, cycles, solvent loss
Integrated sequential leaching	Increases concentration and controls impurities in stages	Greater circuit complexity	Low-grade tailings with dominant impurities	Stage-specific concentration, overall balance, cumulative consumption

4.8. Quantitative Comparison of Published Performance

Qualitative comparisons between technologies obscure the fact that published results derive from feeds whose grades differ by two to three orders of magnitude. Presenting numerical data in a single table requires explicit identification of what parameters were measured, on which material, and at what scale, and quickly reveals which variables remain unreported.

Table 7 compiles the most frequently cited quantitative results for tailings from the past decade, organized by matrix and primary route. Qualitative entries correspond to variables not reported in the original studies; that absence is itself a finding.

Table 7. Published quantitative results for REE recovery from tailings, by matrix and primary route.

Matrix and Feed Grade	Primary Route	Recovery and Product Grade	Selectivity or Dissolved Mass	Scale	Reference
Tin tailings, 3.4% TREO	Carrier-assisted magnetic separation (sodium oleate and emulsified kerosene)	80% TREO recovery; grade increased from 3.4% to 9.0% and to 23.2% after carrier removal	No leaching; no mass dissolved	Bench scale	[56]
Fe-rich tailings pretreated with KOH	Alkaline roasting and ChCl-pTSA leaching	Approximately 95% of total REEs	Approximately 94% of the mass dissolved	Bench scale	[31]

Table 7. *Cont.*

Matrix and Feed Grade	Primary Route	Recovery and Product Grade	Selectivity or Dissolved Mass	Scale	Reference
Porphyry copper tailings, 61.74 mg/kg LREEs and 6.8 mg/kg HREEs	Deep eutectic solvents (ChCl-pTSA; ChCl–lactic acid)	Approximately 44% overall recovery	Not reported	Bench scale	[26]
Uranium tailings	Water leaching of secondary phases	Potential inventory exceeding 10,000 t REEs	Low intensity; U and Th mobilized into the same solution	Bench scale	[28]
Simulated tailings concentrate, 2.4% total REEs	Sequential leaching with pH control, solvent extraction (Cyanex 572, TBP, D2EHPA), and oxalate precipitation	Solids containing approximately 92% HREEs and 92.8% LREEs; REE concentration doubled	Impurities reduced to approximately half those in single-stage leaching	Bench scale	[76]
Coarse coal refuse	Flotation, activation, leaching, and staged precipitation	REE concentrates from liquor; co-recovery of Co, Cr, Li, Mn, Sr, and V after calcination	Requires dedicated Th and U separation	Bench and pilot scale	[52,54,55,123]
Gold tailings	Oxalic acid pretreatment and bioleaching	Increased Pr, Ce, and Eu recovery	Prior Fe removal required	Bench scale	[75]
Finnish carbonatite tailings	Separation by density and magnetic susceptibility	Significant, size-dependent REE losses	Insufficient physical contrast in fines	Bench scale	[30]
Bayan Obo tailings	Preconcentration and flotation in a multi-product circuit	Substantial increase in REO grade	Co-recovery of Fe, Nb, Sc, and F	Plant and bench scale	[123]

Three reporting omissions recur. Almost no study reports reagent consumption per kilogram of REEs recovered, very few close the Th and U mass balance between product and residue, and virtually none report the number of organic-phase or resin cycles using real solutions. Until these three variables are published systematically, comparisons across routes will remain comparisons between experimental conditions rather than processing alternatives.

The same information can be displayed graphically to permit closer comparison than a table. Figure 3 plots each case by feed grade and overall recovery, with symbol size proportional to dissolved mass or rejected mass. Consequently, a high-recovery, low-selectivity outcome is visually distinguished from a moderate-recovery result with low bulk mass dissolution.

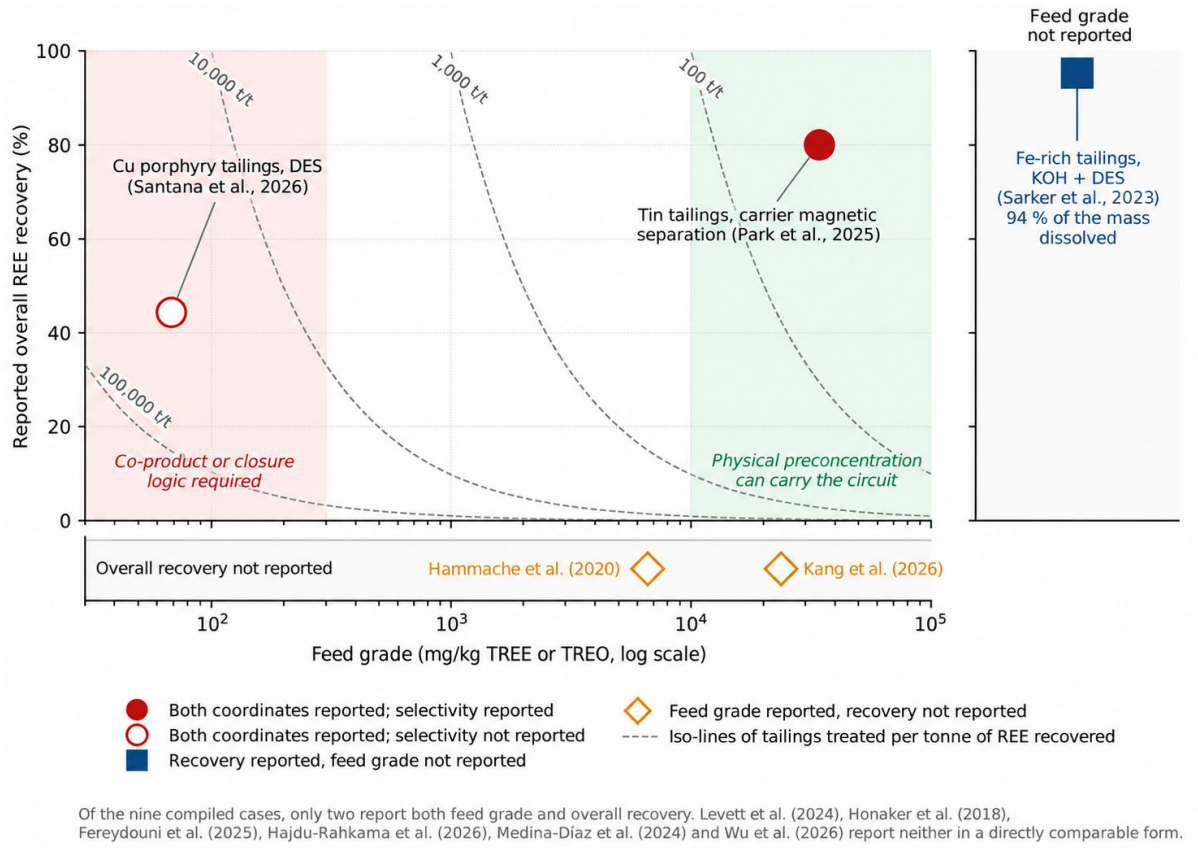


Figure 3. Grade, recovery, and selectivity mapping of published case studies on REE recovery from mine tailings [22,26,28,30,31,55,56,62,69,75,76].

5. From PLS to REE Product

5.1. PLS Quality and Impurity Removal

The REE concentration in the PLS determines the size of the purification circuit. In low-grade mine tailings, a high extraction percentage may still yield solutions with REE concentrations on the order of mg/L and Fe, Al, Ca, Mg, or P concentrations on the order of g/L. This increases the number of stages, extractant consumption, organic-phase losses, and unwanted precipitation. The leaching objective must therefore be framed as maximizing a value function combining recovery, concentration, and selectivity rather than maximizing bulk dissolution.

The sequential leaching developed by Kang et al. for a simulated tailings concentrate containing 2.4% TREE illustrates this principle: pH control approximately doubled REE concentration and reduced impurities to about half those obtained by single-stage leaching. Cyanex 572, TBP, and D2EHPA subsequently separated HREEs, Fe, and LREEs, and oxalate precipitation yielded solids grading ~92% HREEs and ~92.8% LREEs [76]. Although the feed was simulated and richer than many actual tailings, the study demonstrates that PLS engineering can be as important as initial extraction.

PLS quality must be defined before selecting the leaching route. A technically attractive PLS concentrates REEs within a limited volume, maintains a low Fe + Al + Ca + Mg + P/REE ratio, and suppresses problematic F, Th, and U concentrations. For low-grade tailings, solutions containing only a few mg/L of REEs may be unsuitable for conventional SX without a concentration step. Countercurrent leaching, solution recycling, selective adsorption, or membranes may turn modest extraction into a more efficient circuit.

5.2. Solvent Extraction, Ion Exchange, Membranes, and Adsorption

Solvent extraction remains the benchmark technology for large-scale REE separation. Organophosphorus extractants such as D2EHPA, Cyanex 272/572, and solvating systems can separate REE groups and remove impurities, although the chemical similarities among lanthanides require multiple stages and strict acidity control [25,114–116,124]. ILs may act as diluents or functional extractants and modify selectivity, but stability and losses to the aqueous phase must be incorporated in the balance [94,114,115,120,121].

Ion exchange (IX) is particularly attractive for dilute solutions or as a polishing stage. Chelating resins, solvent-impregnated resins, and functional materials can concentrate REEs from sulfate and chloride matrices [81,125–130]. Supported liquid membranes, polymer inclusion membranes, and nanofiltration offer process intensification and lower organic-phase inventories, but stability, fouling, and carrier losses remain limitations [128,131–134]. Adsorption using functionalized carbons, silica, zeolites, MOFs/COFs, and biomolecules is emerging as an alternative for dilute streams; industrial deployment will hinge on selectivity in authentic process matrices and regeneration over multiple cycles [81,125,126].

Membrane technologies are advancing from general concentration toward ligand-assisted separations, nanofiltration, and hybrid processes. The recent literature highlights their value as preconcentration or polishing stages rather than universal replacements for SX [134]. For every technology, tests with synthetic solutions must be complemented by trials using authentic PLS to assess fouling, ionic competition, sulfate/chloride accumulation, and stability over multiple cycles.

5.3. Precipitation, Calcination, and Product Specifications

Oxalate, carbonate, and hydroxide precipitation are common routes for recovering REEs from purified solutions. Oxalate precipitation followed by calcination can yield mixed oxides at high recovery; however, Ca, Th, or U co-precipitation and filterability must be controlled [47,48,61,75,135–137]. When the aim is to sell a mixed carbonate or oxide, individual lanthanide separation may be outsourced; conversely, if value resides in Nd-Pr or HREEs, the plant must accommodate greater complexity. Recent reviews on dysprosium highlight how many separation stages may be required when targeting a high-value HREE from secondary sources [138]. This commercial decision must be established prior to flowsheet selection, as it dictates purity, SX/IX stages, and impurity tolerance.

Oxalate precipitation remains a robust route for mixed REE concentrates and can be integrated with calcination to REOs. Nevertheless, required purity is governed by the market: a mixed carbonate or oxalate may be saleable to a refinery, whereas a magnet supply chain requires individual separation, metallization, and much stricter specifications. The intended final product must therefore be defined in relation to regional refining infrastructure rather than solely on the mineralogy of the tailings [11,139].

Table 8 summarizes the downstream operations applicable to solutions derived from tailings, including their function, the typical interferences in these matrices, and the performance criteria used to evaluate them. Because Fe, Al, Ca, Mg, Th, and U are the dominant interferences across these operations, the performance of each downstream stage depends largely on the selectivity achieved upstream.

Table 8. Downstream operations and critical variables for solutions derived from tailings.

Operation	Function	Typical Interferences	Performance Criterion
Selective neutralization	Remove Fe/Al and adjust speciation	REE co-precipitation; gypsum/hydroxide generation	REE loss below target and substantial Fe/Al reduction
Solvent extraction	Separate LREEs/HREEs or individual REEs	Fe, Ca, high ionic strength, emulsions	Separation factor, number of stages, extractant loss
Ion exchange/resins	Selective capture from dilute PLS	Ca/Mg/Fe competition and site saturation	Capacity, selectivity, regenerability, cycles
Membranes	Concentrate or fractionate before/after SX	Fouling, pressure, salt rejection	Water/REE recovery, concentration factor, energy
Selective adsorption	Capture Sc/Y/HREEs or other targets	Colloidal matter and ionic competition	mg/g, selectivity, desorption, and stability
Oxalic/carbonate precipitation	Obtain a concentrated product	Ca/Th/U co-precipitation	Yield, purity, filterability, and final REO quality

6. Environmental Challenges in Reprocessing REE-Bearing Tailings

6.1. The Dual Nature of Tailings as Resources and Liabilities

Tailings deposits are both mineral inventories and liabilities that pose risks. Reprocessing may reduce volume, remove sulfides or mobile metals, and facilitate better-engineered closure; it may also reactivate surfaces, increase oxidation, generate saline solutions, and alter the physical stability of the deposit. The Global Industry Standard on Tailings Management mandates an integrated knowledge base and management throughout the entire life cycle. Consequently, any remining proposal must be integrated into the safety management of the deposit rather than evaluated as an isolated metallurgical plant [140].

6.2. Acid Drainage, Metals, and REE Mobility

Exposure of residual sulfides during excavation and reprocessing may accelerate acid drainage and mobilize metals. pH strongly controls REE retention in oxides, clays, and secondary material, so a circuit that acidifies the material may alter environmental mobility even if metallurgical recovery is incomplete. The process flowsheet must include acid–base accounting, kinetic tests where applicable, and characterization of leachates from the final residue. Such evaluation is particularly important for historical polymetallic tailings containing coexisting As, Pb, Zn, Cu, or U [32,46].

6.3. Thorium, Uranium, and Radiological Risk

REE recovery may concentrate Th/U in a solid stream, mobilize them into the PLS, or redistribute them among residues. The safest option depends on local regulation and the product. For monazite, early Th/U separation may be required before producing a commercial concentrate; for uranium tailings, recovery of U as a co-product may benefit the project, but radiological control must be demonstrated for effluents and residues [28,45,49]. Reporting REEs alone without radionuclide balances is insufficient to assess actual sustainability.

The benchmark regulatory criterion is numerical and should be incorporated from the flowsheet design stage. International basic safety standards treat any process material as a planned exposure situation if the activity concentration of a radionuclide in the uranium or thorium decay chains exceeds 1 Bq/g, or that of ^{40}K exceeds 10 Bq/g [141]. Tailings may fall below this threshold yet generate streams that exceed it, because concentrating monazite into a small product increases specific activity even as volume decreases. Th and U balances must therefore be expressed as activity per stream (feed, preconcentrate, PLS, product, and residue) rather than solely as percentage removal. Consequently, radiological classification must precede route selection rather than follow from it [56].

Recovering monazite and other phosphates may concentrate Th in a small solid fraction, which may be advantageous in volume terms but increases specific activity and regulatory requirements. At Bayan Obo, U mobility in groundwater downstream of tailings remains a subject of recent research [142]. The appropriate strategy is not to dilute radionuclides into new streams, but to define destinations through the mass balance, product, stabilized residue, or a dedicated management stream.

6.4. Reagents, Water, Energy, and Secondary Salts

An environmental advantage from reprocessing tailings cannot be assumed. The material has already been mined but may still require pumping, excavation, dewatering, re-grinding, roasting, and massive reagent volumes. Life cycle assessments of REE production identify mineralogy, thermal energy, acid, neutralization, and residue disposal as recurring environmental hotspots [140,143–147]. A tailings project may avoid some primary mining impacts but introduces an additional burden associated with disturbing a legacy deposit and managing pore water, acid drainage, or contaminants.

The environmental burden depends on the total mass subjected to chemical processing. This relationship explains why moderate preconcentration may reduce impacts more than replacing an acid with an alternative solvent. Energy demand dominates acid baking and roasting; reagent and neutralizing-agent consumption dominate direct leaching; DES and IL impacts may arise mainly from solvent synthesis and losses; and bioleaching reduces chemical severity but requires time, aeration, and a carbon source. Comparisons must be expressed per kilogram of marketable REO or REEs, not solely per tonne of tailings [143,144].

6.5. Post-Extraction Residues and Risk Transfer

REE extraction modifies and alters mineralogy and specific surface area. Partially degraded silicates, gypsum, soluble salts, Fe-Al hydroxides, and radionuclide-bearing precipitates may alter filterability, geotechnical stability, and leachability. A route can be deemed environmentally favorable only if the final stream is characterized as rigorously as the feed. Leach testing, secondary mineralogy, acid potential, strength, and durability must be assessed before proposing reuse as construction material or backfill [16,19].

Table 9 summarizes the main environmental challenges of reprocessing REE-bearing tailings, together with the corresponding risk mechanism, the indicators required to quantify each risk, and the design measures that can mitigate it. Most of these risks are introduced or amplified by the selected flowsheet and are not intrinsic to the deposit; they must therefore be addressed during process design rather than after it.

Table 9. Environmental challenges and required controls in reprocessing REE-bearing tailings.

Challenge	Risk Mechanism	Indicators	Design Measure
Th and U	Concentration or mobilization during leaching/separation	Mass balance, activity, concentrations in PLS and solids	Stream segregation, targeted precipitation/control, traceability
Acid drainage	Sulfide oxidation during re-excavation or regrinding	NAG/ABA, pH, sulfate, Fe, and metals	Sulfide removal, encapsulation, or engineered neutralization
Water consumption	Washing, thickening, and purification	m^3/t and m^3/kg REO	Countercurrent operation, recycling, and water recovery
Reagents and neutralization	Gangue dissolution and salt generation	kg reagent/t; kg salt/kg REO	Preconcentration, recycling, selective leaching
Energy	Grinding, roasting, drying, pumping	kWh/t and kWh/kg REO	Avoid unnecessary regrinding; heat recovery
Salts and effluents	Sulfates, chlorides, nitrates, Na/K, and precipitates	TDS, ionic load, bleed volume	Partially closed circuits, crystallization/salt recovery
Final residue	Changes in mineralogy, fineness, and leachability	Leaching, mineralogy, geotechnical properties	Design final disposal from the outset and verify performance

7. Circular Economy Opportunities

7.1. Remining and Rehabilitation as a Single Project

The circular economy in tailings involves more than extracting an additional metal. Its greatest potential value arises when remining removes material from a high-risk facility, recovers multiple components, and reconstructs the deposit or site with safer geometry and mineralogy. REE revenue may subsidize part of closure, while avoided rehabilitation costs may improve project economics. This approach is particularly relevant for abandoned deposits and facilities where reprocessing reduces volume or removes acid-generating phases [74,146].

7.2. Co-Products and Multielement Recovery

Recovering Fe at Bayan Obo, P from apatite tailings, or U from uranium tailings shows that product sequencing changes process chemistry [63,75,85,148–153]. By contrast, deliberately dissolving the entire matrix and then restoring selectivity through SX is generally inefficient at low REE grades. A useful design rule is to minimize "non-target mass dissolved per unit mass of REEs recovered." This parameter integrates mineralogy, selectivity, and downstream burden, and can be compared across routes even when initial grades differ.

Co-recovery redefines the effective REE cut-off grade. Fe, Nb, Sc, P, F, Cu, Zn, or even construction materials may contribute value, while removing some of these components simultaneously reduces consumption and risk. Bayan Obo is the clearest example of this

multi-product rationale, but the principle is general: tailings must be modeled as a system of value-bearing streams and penalties rather than as a single-metal ore [60].

7.3. Reuse of Mineral Residues

After REEs and co-products are recovered, the bulk fraction generally remains silicate-, aluminosilicate-, iron-, or carbonate-rich. Geopolymers, aggregates, ceramics, cemented backfill, and cover materials are potential options, but only where residues meet mechanical and environmental performance standards. The literature on integrated tailings utilization emphasizes that residue valorization may be as important as metal recovery for achieving high rates of mass utilization [16].

The sequence matters. If the residue is intended for alkaline activation as a geopolymer, prior leaching may remove interfering species or, conversely, consume Al and modify reactivity. Similarly, acid baking may generate sulfates that restrict construction applications. The reuse route must therefore be evaluated alongside REE extraction rather than post-flowsheet finalization.

7.4. Reagent Recycling and Shared Infrastructure

Internal process circularity requires recovery of water, acid, extractants, resins, and precipitation reagents where feasible. In active mining districts, existing infrastructure (power, thickeners, laboratories, roads, water plants, and smelters/refineries) may reduce capital barriers. Industrial symbiosis also offers potential: acids or bases from other operations, CO_2 for carbonation, waste heat for drying, and cementitious matrices to stabilize the final fraction. These opportunities must be verified through actual balances rather than assumed as theoretical environmental credits.

Table 10 summarizes the circular economy opportunities associated with REE recovery from tailings, the value generated by each, the technical condition required to realize it, and the main risk of overestimating its benefit. In every case, the benefit becomes credible only when it is supported by measured balances, a demonstrated market, or verifiable avoided costs.

Table 10. Circular economy opportunities associated with REE recovery from tailings.

Opportunity	Value Generated	Technical Condition	Risk of Overestimation
REE recovery	Diversifies supply and generates a critical material product	Inventory and flowsheet with selectivity and a saleable product	Using TREE as a proxy for value without individual distributions
Fe/Nb/Sc/P/F/base metal co-products	Increases revenue and reduces impurities	Compatible sequential separation	Assigning value to co-products without a demonstrated market or purity
Tailings volume reduction	Smaller footprint and potential closure improvements	High fraction of mass reused or removed	Transferring the same risk to another facility
Geopolymers/aggregates/backfill	Valorization of the bulk mineral fraction	Mechanical and environmental compliance	Extrapolating laboratory strength to large volumes
Water/reagent recycling	Reduces consumption and effluents	Controlled impurity accumulation	Assuming closed circuits without bleeds
Rehabilitation funded by remining	Integrates recovery and liability reduction	Closure plan, permits, and verifiable avoided costs	Accounting for environmental benefits without a baseline
Industrial symbiosis	Shares energy, reagents, or infrastructure	Proximity and stream compatibility	Hypothetical credits without contracts/logistics

8. Techno-Economic and Environmental Assessment and Decision Framework

8.1. From Percentage Recovery to Net Value per Tonne

Tailings economics differ structurally from primary mining. Reprocessing may bypass exploration, primary crushing, and some mine development, but requires handling fine, potentially saturated and heterogeneous material. Dredging/excavation, thickening, internal transport, reagent, disposal, and rehabilitation costs are site-specific. Analysis of legacy tailings treated by phytoextraction shows that a technically feasible route may remain economically weak if metal productivity is low [21,22]. Conversely, high-grade tailings or those yielding co-products may support more complex circuits.

Project economics can be expressed through a chain of recovery factors. Gross in situ value is calculated from individual grades and prices; metallurgically recoverable value accounts for losses; operating margin deducts mining/re-excavation, preparation, reagents, energy, water, separation, and disposal costs; and integrated value incorporates co-products and avoided rehabilitation costs. Particular attention must be given to price sensitivity for Nd-Pr-Dy-Tb and Sc, as minor variations in their proportions can alter the economic contribution of otherwise similar TREE grades.

8.2. LCA and System Boundaries

Life cycle assessment (LCA) must avoid granting automatic credits for treating "waste." System boundaries must include tailings rehandling, drying or pumping, additional comminution, reagents, thermal energy, neutralization, solvents, water treatment, and final management of the residue. Rare-earth LCA studies reveal that waste management and radioactivity have historically been underrepresented, so secondary systems must explicitly document these streams [143,144].

The study of abandoned tailings by Muñoz-Morales et al. [21] shows that a route may be unprofitable when only the value of recovered REEs is considered, yet remain relevant within a remediation strategy. Comparing projects requires at least two scenarios: recovery as a stand-alone mining project and recovery integrated into a mandatory or closure program.

8.3. Gate-Based Prioritization Framework

Prioritization can be organized around eight gates. Progression requires a minimum level of evidence at each gate, allowing studies with a low probability of success to be discontinued early. This approach avoids investment in leaching optimization before a deposit has a reliable inventory or when mineralogy indicates that value is encapsulated in a fraction that cannot reasonably be concentrated. Figure 4 represents this sequence as a funnel in which the minimum evidence required increases from one gate to the next and in which any gate may lead to terminating or reformulating the study.

Table 11 specifies, for each of the eight gates, the progression question that must be answered and the minimum evidence required to advance to the next stage.

Table 11. Decision framework for prioritizing tailings with REE potential.

Gate	Progression Question	Minimum Evidence
1. Inventory	Are tonnage, geometry, moisture, and variability reliably established?	Domain-based 3D model, density, grade coefficient of variation, and uncertainty
2. Value distribution	Does the Nd-Pr-Dy-Tb-Sc-Y distribution justify the study?	Individual grades, TREO, Th/U, and preliminary value

Table 11. *Cont.*

Gate	Progression Question	Minimum Evidence
3. Mineralogy	Are hosts, associations, and liberation acknowledged?	Quantitative mineralogy, deportment, and grain size
4. Mass reduction	Can gangue be rejected before intensive chemical processing?	Enrichment factor, recovery, and rejected mass
5. Leaching	Does the PLS concentrate REEs with controlled co-dissolution?	REEs/kg reagent, PLS concentration, dissolved Fe-Al-Ca-Mg-P
6. Downstream	Can a saleable product be obtained?	Separation factor, purity, SX/IX/membrane cycles, and losses
7. Environment	Does the process reduce liabilities rather than transfer them?	Th/U, water, energy, and salt balances; leachability and geotechnical properties
8. Economics and closure	Is there a margin under conservative assumptions and a defined final destination?	Techno-economic assessment, sensitivity, co-products, avoided costs, and closure plan

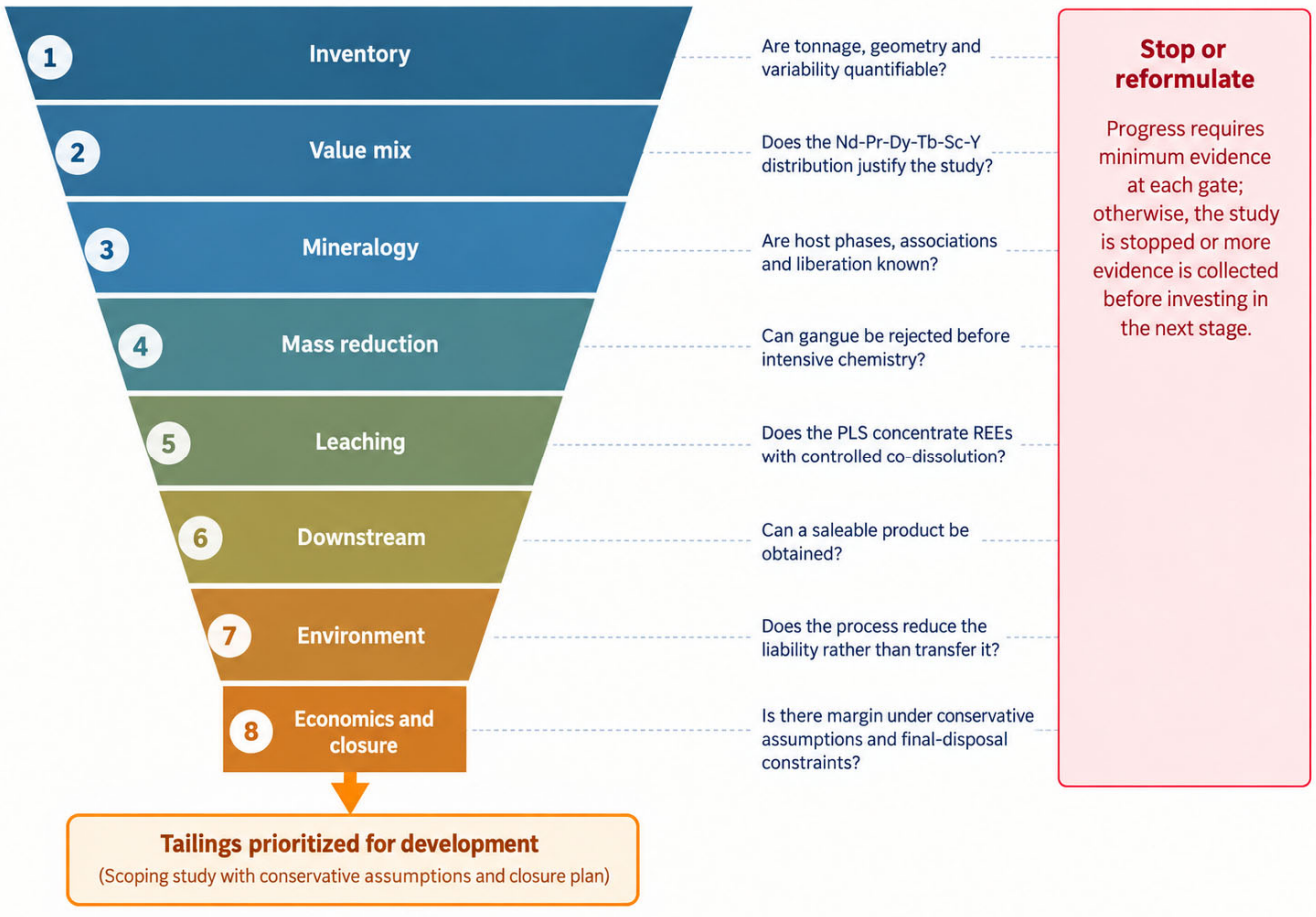


Figure 4. Gate-based prioritization funnel: Progression from one gate to the next requires increasing minimum evidence, and any gate may lead to terminating or reformulating the study before investment in the next stage.

8.4. Economic Threshold: Contained Value per Tonne Treated

A central question in assessing tailings economics is: What is the maximum value of the tonne to be treated? Gross in situ value is the sum of individual grades multiplied by their oxide prices, and its magnitude depends much more on the proportions of Nd, Pr, Dy, and Tb than on total REEs, given that La and Ce oxides trade at prices two orders of magnitude below those of magnet-related oxides.

Table 12 applies this calculation to the same reference grades using two contrasting compositions: an LREE-dominated mixture containing 75% La and Ce, 20% Nd and Pr, and 0.5% Dy and Tb, and a mixture enriched in magnet elements containing 55% La and Ce, 35% Nd and Pr, and 2% Dy and Tb. Market pricing corresponds to September 2026 (approximately US$1 and US$3 per kilogram for La and Ce oxides, US$100 for Nd-Pr oxide, US$230 for Dy oxide, and US$1100 for Tb oxide) and is extremely volatile. The exercise should therefore be read as an indication of orders of magnitude and updated using prices prevailing at the time of each assessment.

Table 12. Gross in situ value of one tonne of tailings by grade and elemental composition, and realizable value at a combined recovery and payability factor of 20%.

Reference Grade (mg/kg REE Oxides)	Associated Family	In Situ Value, LREE-Dominated Mixture (US$/t)	In Situ Value, Mixture Enriched in Magnet Elements (US$/t)	Realizable Value at 20% (US$/t)
70	Copper porphyries	1.7	3.3	0.3–0.7
300	Abandoned Pb-Zn tailings	7	14	1.5–2.9
1000	Coal-derived residues and low-grade tailings	24	48	5–10
6400	Phosphate residues and carbonatites	156	305	31–61
34,000	Tin tailings	830	1620	166–324

Authors' calculations. September 2026 market prices, subject to high volatility. Realizable value assumes 40% overall recovery and payment for half the contained value of a mixed product; combined, these factors equal 20% of in situ value.

This exercise establishes priorities more clearly than qualitative discussion alone. Below approximately US$20 per tonne of realizable value, a complete hydrometallurgical circuit applied to the entire mass is unlikely to be viable independently, since rehandling, dewatering, reagents, neutralization, and residue disposal incur costs of that order. Copper porphyry and abandoned Pb-Zn tailings fall within this range, where projects are justified only when REEs are recovered as co-products of other reprocessing operations or as components of closure programs that account for avoided costs. Between US$20 and US$100 per tonne, viability depends on mass reduction: if preconcentration captures value in a small fraction, the chemical stage treats a feed equivalent to medium-grade ore. Above this range, the challenge shifts from contained value to selectivity, product specifications, and Th and U management. Sensitivity to the Nd-Pr oxide price dominates across all ranges: a 30% change in that price affects the value per tonne more than doubling La and Ce grades.

The result can be modeled as a continuous relationship that is more informative than a five-row table. Figure 5 plots realizable value per tonne against REE grade on a logarithmic scale, with a band between the two elemental mixtures and a horizontal band indicating the treatment cost range. Their intersection graphically defines the cut-off grade for each family. The treatment cost band (approximately US$10–100 per tonne treated) includes only indicative operating costs (OPEX) for rehandling, dewatering, reagents, neutralization, and residue disposal; it excludes capital cost recovery (CAPEX) and long-term closure

or rehabilitation costs, whose inclusion would raise the band and shift the cut-off grade upward, and it does not credit avoided closure costs.

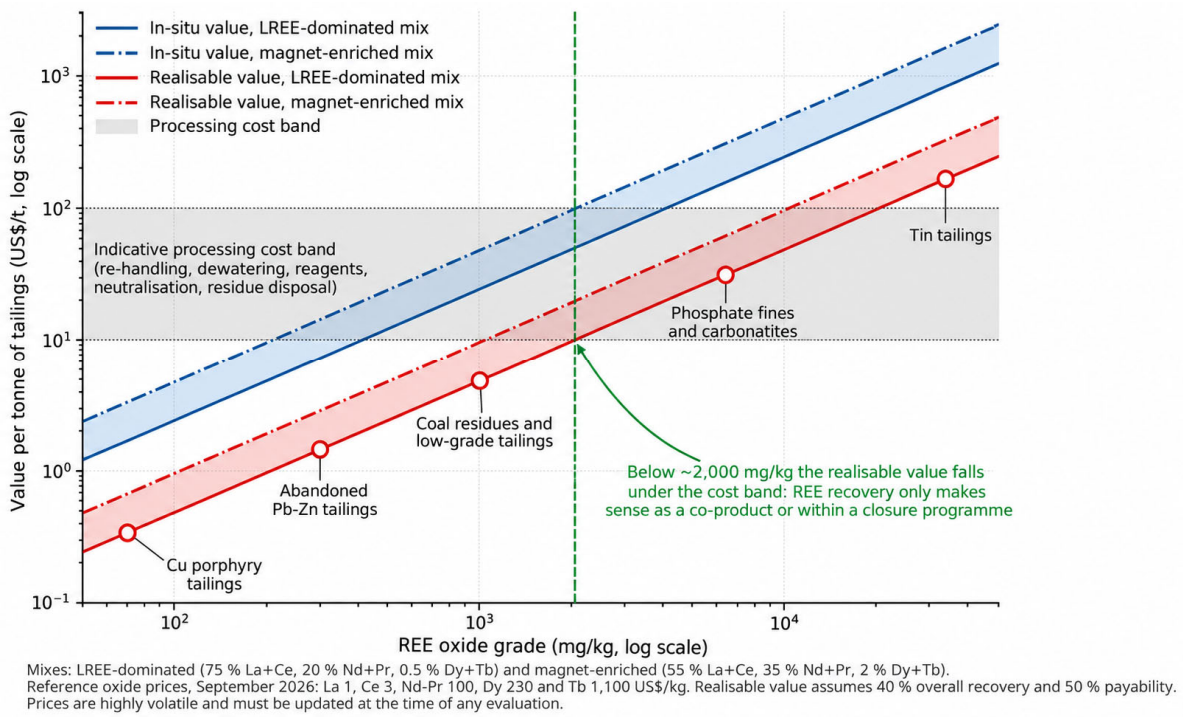


Figure 5. Realizable value per tonne of tailings versus REE grade, incorporating elemental composition and treatment cost bands. The treatment cost band represents indicative operating costs (OPEX) only; capital (CAPEX) and closure costs are excluded.

8.5. Recommended Reporting Checklist for E3–E5 Studies

The reporting omissions revealed by Table 7 (reagent consumption per kilogram of REEs recovered, Th and U mass balances, and the number of separation cycles completed with real solutions) prevent published results from being compared on a common basis. Table 13 proposes a minimum reporting checklist organized according to the evidence levels defined in Table 3, specifying the information that becomes mandatory once a study claims to reach E3, E4, or E5. The checklist is cumulative: a study claiming E4 must also report all E3 items, and a study claiming E5 must report all E3 and E4 items. Its systematic adoption would allow cases to be compared on the basis of recoverable value and liability reduction rather than experimental conditions, and would supply the evidence required at each gate of Table 11.

Table 13. Recommended minimum reporting checklist for studies on REE recovery from mine tailings, by evidence level.

Reporting Block	Minimum Information to Report	Required From
Sampling and inventory	Sampling design (domains, depth, and number of samples), dry tonnage, density, and representativeness of composites	E3
Head assay	Individual REEs, Sc, and Y; TREE and TREO with the oxide conversion used; Th, U, Fe, Al, Ca, Mg, P, F, Si, and S; digestion and analytical method	E3

Table 13. *Cont.*

Reporting Block	Minimum Information to Report	Required From
Mineralogy	REE host phases, size-by-size deportment, liberation, and mineral associations	E3
Test conditions and recovery	Particle size, reagent type and dose, solid-to-liquid ratio, temperature, time, and pH; element-specific recovery calculated from both solution and residue assays, with replicates	E3
Reagent consumption and PLS quality	Reagent consumption per tonne treated and per kilogram of REEs recovered; REE and impurity concentrations in the PLS; non-target mass dissolved per unit mass of REEs extracted	E3
Mass and radionuclide balances	Closed balances of mass, REEs, Th, and U by stream (feed, preconcentrate, PLS, product, and residue), with Th and U also reported as activity concentration (Bq/g)	E4
Separation and product	Separation factors; number of SX, IX, or membrane cycles completed with real solutions; extractant or resin losses; and product purity and impurity profile	E4
Residue, water, and energy	Residue mineralogy, leachability, acid generation potential, and geotechnical properties; water (m^3/t) and energy (kWh/t) consumption	E4
Continuous operation and assessment	Campaign duration, throughput, steady-state variability, and reagent recycling; techno-economic and life cycle assessments with stated system boundaries and price basis	E5

Source: Authors' own elaboration. Items are cumulative: studies claiming E4 or E5 must also report all items required at lower levels.

9. Conclusions

Mine tailings are a globally distributed secondary REE source, yet they do not constitute a uniform resource class. Their potential ranges from polymetallic streams with high residual grades to very-low-concentration deposits where value emerges only through large tonnage, co-products, or rehabilitation. Evaluation must distinguish geochemical content, potential inventory, technical resource, and recoverable inventory.

Mineralogy and mass reduction dominate technology selection. When REEs are liberated or partitioned by size, physical separation can transform the economics of chemical stages. When they are encapsulated in monazite, bastnaesite, xenotime, or silicates, thermal, alkaline, or mechanochemical activation may be required. In low-grade tailings, selectivity and PLS concentration are more important than maximizing percentage extraction.

Recovery does not end with leaching. Fe, Al, Ca, Mg, P, F, Th, and U determine purification costs, environmental stability, and product specifications. SX, IX, membranes, adsorption, and precipitation must be evaluated using real solutions over multiple cycles. Studies from 2024 to 2026 show advances in sequential leaching, ultrafine preconcentration, DESs, and biomining; however, most systems still require validation in continuous pilot-scale operation.

The strongest circular economy opportunity arises when REE recovery is combined with co-products, reduced tailings volume, water and reagent recycling, safe residue reuse, and rehabilitation. A flowsheet should not be classified as circular merely due to the secondary origin of its feed; rather, it must demonstrate enhancement in resource recovery and reductions in the system's total liability. The gate framework helps prioritize deposits for which the inventory, mineralogy, selectivity, product, environmental performance, and closure plan are compatible under conservative economic assumptions.

Mass and value calculations define the problem more clearly than any list of technologies. Recovering one tonne of REEs requires treating approximately 74 t of tin tailings, 390 t of phosphate residue, 2500 t of low-grade tailings, or 36,000 t of copper porphyry tailings at 40% overall recovery. Realizable value per tonne treated varies correspondingly from less than one dollar to several hundred. These extremes represent differences in the nature of the project rather than simply differences in degree: at the low-grade end, REE recovery exists only as a co-product or closure component, while at the high-grade end, the challenge shifts to selectivity, radiological control, and commercial specifications.

Evidence quality remains the dominant limitation. Most of the literature concerns mineralogical characterization and batch metallurgical tests; integrated circuits with closed mass balances remain scarce, and no verified continuous operations exist outside iron-associated multi-product schemes. Three measurements would change the current ranking more substantially than any new reagent: size-specific grade–recovery curves with Nd, Pr, Dy, and Tb deportment in real samples; closed balances reporting Th and U activity by stream alongside reagent consumption per kilogram of product; and continuous campaigns sustaining tens of separation-stage cycles with real solutions. Reporting obligations taking effect in Europe in 2026–2027 will generate, for the first time, a comparable dataset on extractive waste. The rigor of their analytical requirements will determine whether the next decade focuses on recoverable inventories rather than geochemical contents.

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