

Temperature-driven thorium-uranium separation during high pulp density water leaching

Esra Baştürkçü

Department of Mineral Processing Engineering, Faculty of Mines, Istanbul Technical University, 34467, Istanbul, Türkiye

Corresponding author: tanisali@itu.edu.tr (Esra Baştürkçü)

Abstract: Managing radioactivity caused by thorium (Th) and uranium (U) is a critical challenge in rare earth element (REE) hydrometallurgy. Sulfuric acid baking followed by water leaching is one of the most widely adopted industrial routes in REE processing; conventional Th-U separation within this process either relies on high-temperature acid baking ($>300\text{ }^{\circ}\text{C}$) to form insoluble thorium pyrophosphate or requires dedicated downstream purification steps after co-leaching. This study presents an innovative approach to selectively separate Th from U directly within a single water-leaching step by optimizing leaching parameters without high-temperature acid baking. A REE pre-concentrate from the Eskişehir-Beylikova deposit was acid-baked under moderate conditions ($250\text{ }^{\circ}\text{C}$) for 60 min with a 1:1 acid-to-sample ratio and subsequently water-leached under high pulp density ($S/L = 1:2$). The effects of leaching temperature ($25\text{--}90\text{ }^{\circ}\text{C}$) and time ($15\text{--}120\text{ min}$) on Th and U dissolution behavior were systematically investigated. Mineral Liberation Analysis (MLA) confirmed that Th and U were exclusively hosted by distinct mineral phases, Th-parasite and plumbophybrochlore, respectively. The experimental results demonstrated that U dissolution was temperature and time-independent, maintaining a stable extraction of 90% due to soluble uranyl sulfate formation. In contrast, Th recovery sharply declined to $<10\%$ at $90\text{ }^{\circ}\text{C}$ due to the temperature-dependent decrease in stability of Th-sulfate complexes. A recovery ratio (RR) of approximately 13.3 ($\text{RR} = \%U / \%Th\text{ recovery}$) was achieved within the water leaching step alone, demonstrating that Th-U separation can be completed at the leaching stage, thereby reducing process complexity and the number of required unit operations.

Keywords: thorium-uranium separation, water leaching, acid baking, mineral liberation analysis, selective leaching

1. Introduction

Thorium and uranium are radioactive elements of growing strategic interest: uranium remains the principal fuel for conventional nuclear reactors, while thorium is increasingly considered a complementary nuclear fuel cycle owing to its greater natural abundance and the potential safety advantages of thorium-based reactor designs (Pratama et al., 2025). Global thorium resources are estimated at 6.4 million tonnes, while uranium resources total roughly 6.1 million tonnes, with production concentrated in a handful of countries (Kazakhstan, Canada, Namibia, and Australia for uranium) (USGS, 2025).

Uranium is primarily extracted from sandstone-hosted, unconformity-related, and vein-type deposits, with major operations in Kazakhstan, Canada, Namibia, and Australia (USGS, 2025). Thorium, by contrast, occurs principally as monazite, thorianite, and thorite in placer, carbonatite, and vein-type deposits; however, even where these are mined, thorium is most often recovered as a co-product of heavy-mineral-sand (monazite) processing for REE and titanium/zirconium minerals rather than as a standalone target, since there is currently little to no commercial demand for thorium itself. Within REE-bearing deposits, thorium and uranium are typically found together as accessory constituents hosted in the REE minerals themselves. Where they are recovered at all, both are obtained only as by-products of REE ore processing rather than as primary targets, with their recovery and management driven not only by potential economic value but, more importantly, by the need to comply with environmental

regulations governing the handling and disposal of radioactive materials (Kanazawa and Kamitani, 2005; Soldenhoff, 2013; Zhu et al., 2015; USGS, 2025).

The specific mechanism by which thorium and uranium are recovered or rejected as by-products depends strongly on the REE processing route and deposit mineralogy. One of the most widely adopted industrial routes for processing REE-bearing ores is sulfuric acid baking. Demol et al. (2018) showed that sulfation of monazite is essentially complete at 250 °C, giving over 90% solubilization of REE, Th, and phosphate, but above 300 °C, a thorium phosphate-type precipitate forms during leaching, sharply reducing Th extraction while REE recovery stays near-complete, evolving into insoluble thorium-REE polyphosphates at 400-500 °C that retain both elements in the residue. Similarly, at Bayan Obo (China), concentrates digested above 500 °C form insoluble thorium pyrophosphate, rejecting Th to the leach residue while REEs report to the leach liquor; a lower-temperature variant (<300 °C) avoids this, so Th instead reports to solution and must be removed downstream by solvent extraction (Huang et al., 2006; Wang and Wang, 2006; Du et al., 2010). Comparable acid-bake/water-leach routes, with Th partitioning governed by baking temperature, are used at Lynas' Mount Weld (Australia) and Molycorp's Mountain Pass (USA); uranium, given its low concentration, is typically co-rejected with Th or unreported (Zhu et al., 2015).

Beyond the temperature-controlled pyrophosphate route, several other methods are used to separate U and Th from REEs once both are in solution: selective precipitation, solvent extraction (SX), ion exchange/chromatography, and, more recently, membrane-based separation (Carrillo García et al., 2020). Selective precipitation exploits the distinct precipitation pH of trivalent REEs (6.8-8.0), Th (as low as 1-2 in sulfate liquors), and U (5.5-7), as applied at the Nolans and Kvanefjeld projects (Pawlik, 2013; Zhu et al., 2015; Carrillo García et al., 2020). Oxalate precipitation can also co-precipitate REE and Th while leaving U in solution, with up to 98% Th and 99% REE recovery (Amer et al., 2013). SX is the most widely studied route. Primary amines selectively extract Th over REEs (~97% Th at pH 3, ~90% U at pH 4), while tertiary amines favor U; combined, they achieve >99% extraction of both with minimal REE loss (Kanojia et al., 2025; Carrillo García et al., 2020). Acidic organophosphorus extractants (D2EHPA, EHEHP) similarly favor U(VI)/Th(IV) over REE(III) (Sato, 1962, 1968, 1989; Gupta and Singh, 2003; Zhu et al., 2015), and Lan et al. (2025) recently achieved ~100% co-extraction of U and Th using a single extractant (HEHEHP). Greener alternatives such as ionic liquids are also emerging (Kanojia et al., 2025). Ion exchange and membrane methods offer high selectivity but remain limited to small-scale or polishing applications due to low throughput (Carrillo García et al., 2020).

Despite this substantial body of work on Th-U separation, most existing studies address Th and U behavior either as a fixed outcome of the acid-baking temperature or as a subsequent solution-purification step (precipitation, SX, IX) applied after REEs have already been leached into solution. In contrast, this study shows that neither high-temperature acid baking nor an additional separation stage is required to separate Th and U. By optimizing water-leaching conditions alone, Th and U can be separated directly in a single step. In this context, the present study investigates the effect of temperature (25-90 °C) and leaching time (15-120 min) on the dissolution behavior of Th and U during water leaching of an acid-baked REE pre-concentrate under high pulp density conditions (S/L = 1:2), with particular emphasis on the markedly different chemical behavior of the two elements under these conditions. To the authors' knowledge, this is the first study to show that water leaching alone, without high-temperature baking, can achieve early and effective separation of Th and U, offering a simpler, leaching-driven alternative to the precipitation- and SX-based separation routes reported in the literature.

2. Materials and methods

2.1. Material and sample characterization

The feed material used in all experiments was an REE pre-concentrate from the Eskişehir-Beylikova deposit, produced by comminution, mechanical attrition, and classification, as described in Baştürkçü et al. (2022). XRF analysis of the REE pre-concentrate showed that it was predominantly composed of Ca- and Ba-bearing phases together with rare earth element oxides (REO, 9.5%), with Fe₂O₃ (4.5%) and SiO₂ (3.5%) as additional constituents, as shown in Table 1 (Gürkan et al., 2026). Based on the fluorite and barite host phases independently confirmed by MLA (Fig. 2), the Ca-F and Ba-S elemental signals

detected by XRF were assigned stoichiometrically to CaF_2 (40.1%) and BaSO_4 (22.4%), respectively. The pre-concentrate had a particle size distribution with d_{50} : 5 μm and d_{90} : 33 μm , indicating a fine-grained feed material. Particle size distribution was analyzed using a Malvern Mastersizer 2000.

Table 1. XRF results of the preconcentrate produced from Beylikova ore (Gürkan et al., 2026)

Component	CaF_2	BaSO_4	REO	Fe_2O_3	SiO_2	Others
Content (%)	40.1	22.4	9.5	4.5	3.5	20.0

* CaF_2 and BaSO_4 contents were calculated stoichiometrically from XRF elemental data based on the fluorite and barite host phases independently confirmed by MLA (Fig. 2)

To establish a quantitative link between the hydrometallurgical response and the ore mineralogy, Mineral Liberation Analysis (MLA, MTA-Quanta 400 MLA, FEI Company) was utilized to identify the modal mineralogy, elemental distribution in minerals, and MLA-EDS visuals used as input in the present study. To establish a quantitative link between the hydrometallurgical response and the ore mineralogy, Mineral Liberation Analysis (MLA, MTA-Quanta 400 MLA, FEI Company) was utilized to identify the modal mineralogy, elemental distribution in minerals, and MLA-EDS visuals used as input in the present study. Analysis was performed in XBSE (X-ray + backscattered electron) mode, in which particles are first segmented by BSE grey-level contrast and then systematically classified via point EDS analysis. A total of 86,507 particles (118,476 grains) were analyzed. Th and U were assigned exclusively (100%) to Th-parisite and plumbophyrochlore, respectively, based directly on the elemental distribution output of the MLA analysis, in which no other identified mineral phase carried a measurable Th or U signal. Fig. 1 presents the modal mineralogy of the pre-concentrate.

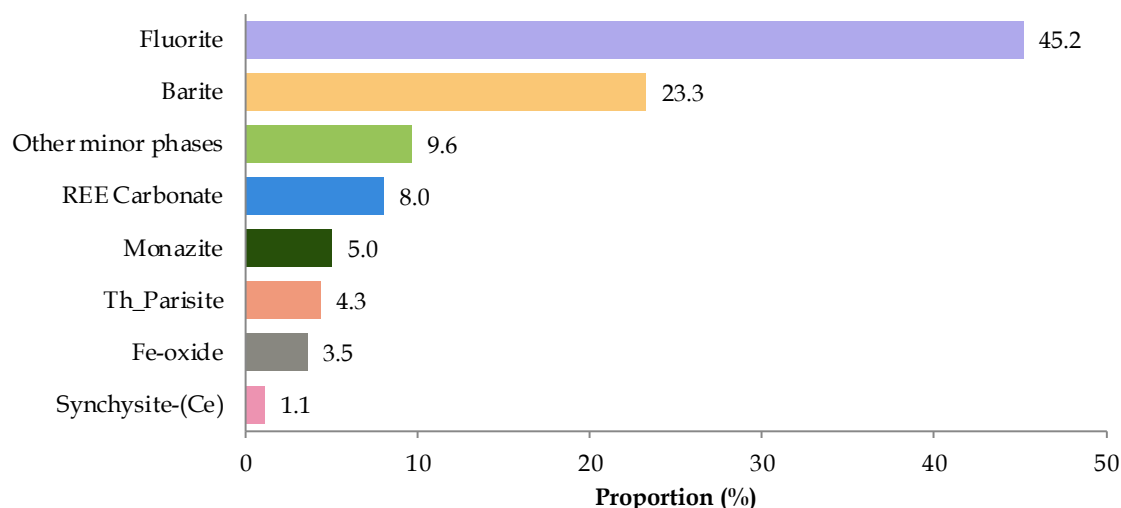


Fig. 1. Modal mineralogy of pre-concentrate

The concentrations of Th and U were determined by inductively coupled plasma mass spectrometry (ICP-MS, Agilent 7800) after lithium borate fusion, which converted the solid samples into a homogeneous melt; the resulting melt was subsequently dissolved in dilute acid to obtain clear solutions suitable for trace element analysis. ICP-MS analysis of the acid-baked concentrate revealed Th-U concentrations in Table 2. Th and U concentrations were determined on the acid-baked sample, as this baked material constitutes the feed to the water leaching step and forms the basis for the recovery calculation (Eq. 1).

Table 2. Th-U content of baked sample (ICP-MS)

Element	Concentration (ppm)
Th	783
U	147

2.2. Acid baking and water leaching procedure

Prior studies have established that H_2SO_4 is the most effective reagent for sulfation of this concentrate, yielding superior REE extraction compared to alternatives (Baştürkçü et al., 2026a). Based on the optimized conditions reported by Baştürkçü et al. (2026b), acid baking was performed using 98% H_2SO_4 (added in pure form) at 250 °C for 60 min with an acid-to-sample ratio of 1:1 (100 g:100 g) in a muffle furnace (Protherm PLF 110-130). Water leaching experiments were conducted on 20 g of baked sample with 40 g of leaching solution (S/L = 1:2) in a jacketed double-walled vessel connected to a chiller unit for precise temperature control, with temperature additionally verified via a thermocouple, agitation provided by a magnetic stirrer at 400 rpm, and pH continuously monitored via a pH electrode, as schematically illustrated in Fig. 2. As leaching was carried out in this closed, jacketed vessel, evaporation and any resulting change in liquid volume were prevented.

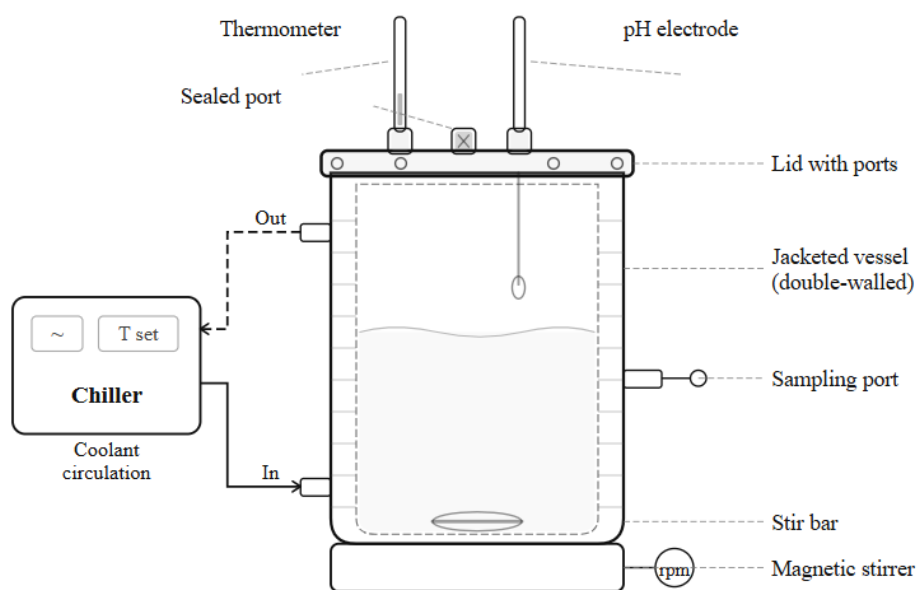


Fig. 2. The visual of the water leaching equipment

Table 3 summarizes the effects of temperature and leaching time on the dissolution behavior of Th and U, along with the experimental parameters. At the end of each leaching test, the slurry was filtered under vacuum, and the leach residue was washed to remove entrained soluble species before drying and analysis.

Metal recoveries were calculated using Eq. (1), based on solid-phase mass balance:

$$R = \left(100 - \frac{W \cdot w}{F \cdot f} \right) \times 100 \quad (1)$$

where R is the recovery (%), W is the mass of the leach residue (g), w is the elemental content of the residue (ppm), F is the mass of the feed (baked sample) (g), and f is the elemental content of the feed (baked sample) (ppm). All leaching experiments were conducted in duplicate, and the results are reported as mean values with a relative deviation of $\pm 2\%$.

Table 3. Variable and constant parameters of the leaching experiment

Process	Parameters	unit	Conditions
Water leaching	S:L ratio	g/cm ³	1:2 (33.3% w/w solids)
	Time	min	15, 30, 60, 120
	Temperature	°C	25, 50, 70, 90
	Stirring speed	rpm	400
Acid Baking	Acid-to-sample ratio		1:1
	Temperature	°C	250
	Time	min	60

3. Results and discussion

3.1. Effect of temperature & time on leaching Th and U

Figs. 3 and 4 show the dissolution behavior of Th and U, as a function of temperature and leaching time. Th and U showed markedly different, element-specific responses to the experimental conditions. For numerical transparency, Tables 4 and 5 summarize the mean Th and U recovery values at each temperature-time combination.

At 25 °C and 50 °C, leaching efficiency increased with time, reaching approximately 67-70% and 65%, respectively, indicating dissolution-dominant behavior at lower temperatures. However, at 70 °C and 90 °C, recoveries declined sharply and progressively with increasing leaching time, falling to approximately 7%. To the authors' knowledge, this temperature- and time-dependent decline under fixed baking conditions has not been previously reported in the literature; existing studies instead attribute Th retention primarily to the baking temperature itself, whereby high-temperature baking (>300 °C) promotes thorium pyrophosphate formation and retains Th in the solid residue, while low-temperature baking converts Th to soluble sulfate forms that dissolve readily upon water leaching (Demol et al., 2018; Zhu et al., 2015). The present study shows that even when Th is initially converted to a soluble sulfate form, leaching temperature significantly and previously unreported influences its subsequent dissolution behavior. This observation is consistent with the thermodynamic complexation behavior of Th^{4+} in sulfate media: Zielen (1959) reported association constants of $K_1 = 166$ and $K_2 = 21.7$ for the stepwise formation of ThSO_4^{2+} and $\text{Th}(\text{SO}_4)_2$, with slightly exothermic heats of complexation ($\Delta H_1 = -0.54$ kcal/mol, $\Delta H_2 = -0.89$ kcal/mol). The exothermic nature of these reactions is consistent with decreased Th-sulfate complex stability, and hence Th solubility, with increasing temperature. However, the specific aqueous species and solid phase responsible for Th retention at high temperature were not directly identified in this study; possible re-precipitation, adsorption, or the formation of a secondary Th-bearing phase cannot be excluded and would require dedicated residue characterization (e.g., XRD or SEM) to confirm, which was beyond the scope of the present work.

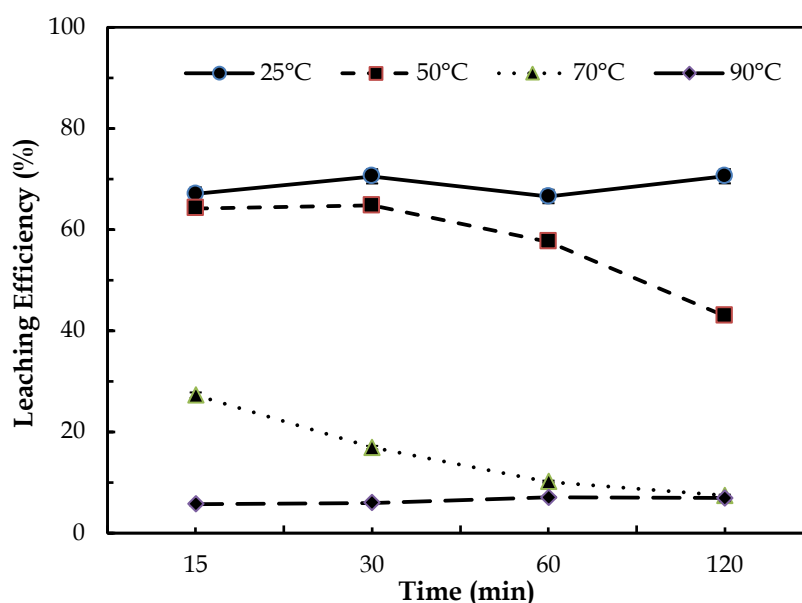


Fig. 3. Variation of leaching efficiency of Th as a function of time and temperature (error bars, $\pm 2\%$ relative deviation, are smaller than the plotted symbols in Fig. 3 and therefore omitted)

U (Fig. 4) exhibited a high and stable leaching efficiency remaining at approximately 90% across all temperatures (25-90 °C) and leaching durations (15-120 min), with no discernible dependence on either variable. This behavior is directly attributable to the sulfation roasting step applied before water leaching, during which uranium oxides are converted to soluble uranyl sulfate (UO_2SO_4) species. The high aqueous solubility of UO_2SO_4 and the thermodynamic stability of the resulting uranyl sulfate complexes $\text{UO}_2\text{SO}_4(\text{aq})$, $\text{UO}_2(\text{SO}_4)_2^{2-}$, and $\text{UO}_2(\text{SO}_4)_3^{4-}$ in acidic media render uranium dissolution highly

favorable during the subsequent water leaching stage (Vercouter et al., 2008). Pratama et al. (2025) similarly demonstrated that low-temperature acid baking of monazite at 200 °C resulted in approximately 100% extraction of Th, U, and P in the subsequent water leach, confirming the near-quantitative conversion of these elements to soluble sulfate forms at low baking temperatures. The temperature- and time-independent extraction profile observed in this study is therefore consistent with complete sulfation of uranium during baking, leaving no kinetic or thermodynamic barrier to dissolution during water leaching.

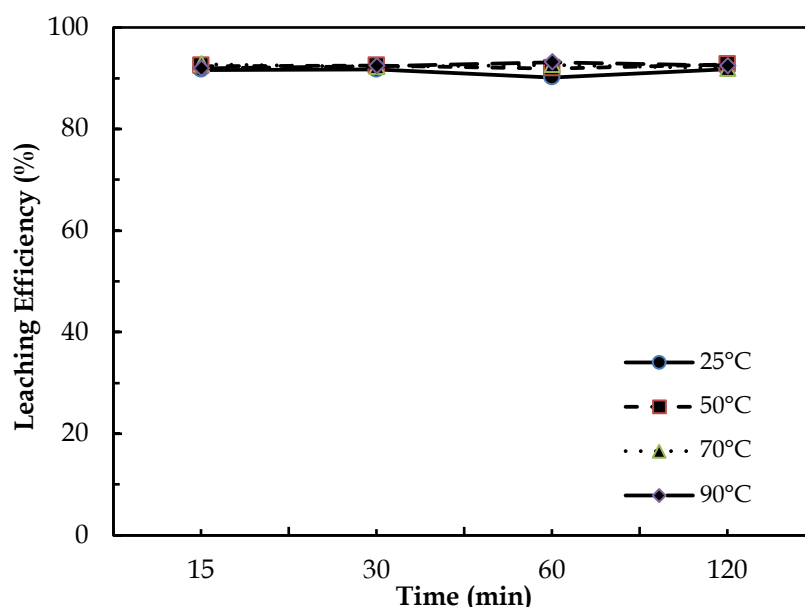


Fig. 4. Variation of leaching efficiency of U as a function of time and temperature (error bars, $\pm 2\%$ relative deviation, are smaller than the plotted symbols in Fig. 4 and therefore omitted)

Table 4. Mean Th recovery (%) as a function of leaching temperature and time (relative deviation between duplicates: $\pm 2\%$)

Time (min)	25 °C	50 °C	70 °C	90 °C
15	67.0	64.2	27.3	5.7
30	70.5	64.8	16.9	6.0
60	66.6	57.6	10.2	7.1
120	70.6	42.9	7.5	7.0

Table 5. Mean U recovery (%) as a function of leaching temperature and time (relative deviation between duplicates: $\pm 2\%$)

Time (min)	25 °C	50 °C	70 °C	90 °C
15	91.7	92.4	92.8	92.0
30	91.7	92.5	92.3	92.4
60	90.1	91.9	92.7	93.1
120	91.8	92.7	92.0	92.5

At the optimum condition (90 °C, 120 min), this divergence corresponds to a recovery ratio (RR = %U recovery / %Th recovery) of approximately 13.3, providing a direct, quantitative measure of the Th-U separation achieved within the water leaching step alone.

3.2. Relationship between mineralogy and leaching efficiency

To rationalize the contrasting dissolution behavior of Th and U observed during water leaching, MLA was used to identify the specific mineral hosts of these two elements in the pre-concentrate. As shown

in Fig. 5, Th and U are each hosted entirely (100%) by a single, distinct mineral phase: Th is exclusively associated with Th-parisite, whereas U is exclusively hosted by plumbophyrochlore. This clean mineralogical separation indicates that the markedly different leaching responses of Th and U cannot be attributed to shared host-mineral behavior, but instead reflect the intrinsically different reactivity of these two carrier phases, a REE-fluorocarbonate (Th-parisite) versus a pyrochlore-group oxide (plumbophyrochlore) toward sulfuric acid during baking and subsequent water leaching.

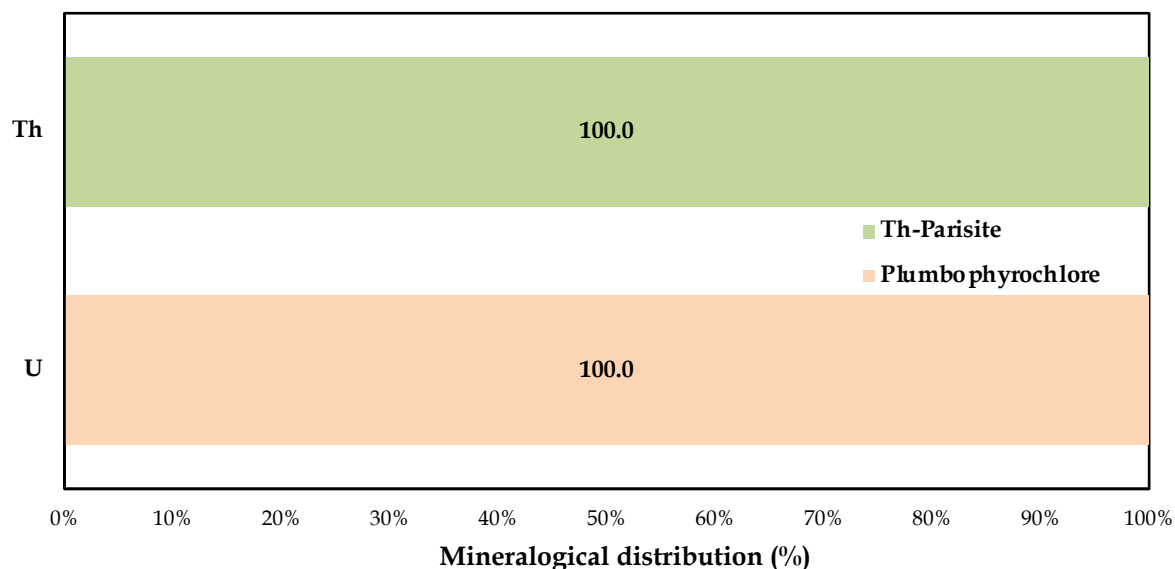


Fig. 5. Mineralogical distribution of Th and U in the pre-concentrate sample

Further insight into this contrast is provided by the cumulative mass recovery curves for mineral liberation by free surface (Fig. 6). Among the gangue and REE-bearing phases evaluated barite, monazite, fluorite, synchysite-(Ce), and Th-parisite, Th-parisite consistently displays the lowest free surface liberation across all liberation classes, with cumulative mass recovery of Th-parisite falls below 20% beyond the 75% liberation class (20.80% at $\leq 70\%$, decreasing to 18.88% at $\leq 75\%$), rather than at the 50% liberation class. This indicates that Th-parisite occurs predominantly as fine, intergrown inclusions with very limited exposed surface area, rather than as liberated or near-liberated grains. Notably, plumbophyrochlore, the host phase of U, could not be evaluated by free surface liberation, as it was not resolved as a discrete liberation class in the MLA dataset, likely due to its low modal abundance and its occurrence as scattered, sub-resolution inclusions within the host particles.

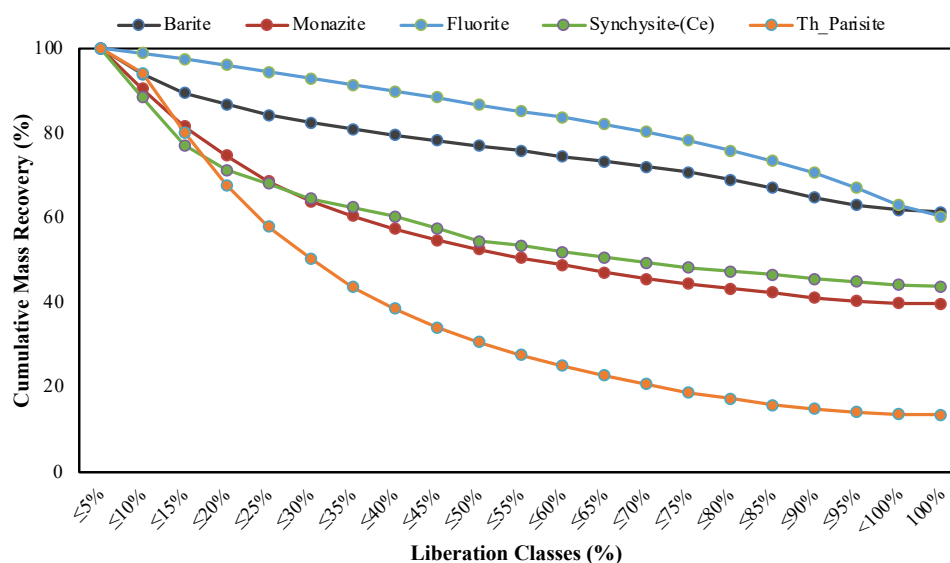


Fig. 6. Cumulative mass recovery curves for mineral liberation by free surface in the pre-concentrate sample

This poorly liberated, fine-grained occurrence of Th-parisite is directly confirmed by the BSE images and corresponding MLA mineral maps in Fig. 7. Across all three representative particles, Th-parisite (red) consistently occurs as irregular, intimately intergrown patches in direct textural contact with fluorite, REE-carbonate, synchysite-(Ce), and the Ba-Mn oxide phase, rather than as isolated, fully enclosed grains. Because fluorite, the REE-carbonates, and the Ba-Mn oxide phase are themselves readily attacked and converted to soluble sulfates during acid baking, their dissolution along these shared grain boundaries effectively opens reaction surface area around the adjoining Th-parisite, allowing acid to penetrate the particle interior. This textural relationship is consistent with the dissolution-dominant Th leaching behavior observed at lower temperatures (Fig. 3): once acid penetrates along these reactive boundaries, Th-parisite becomes accessible to leaching, and its recovery is therefore governed less by its intrinsic liberation than by the reactivity of the surrounding mineral matrix.

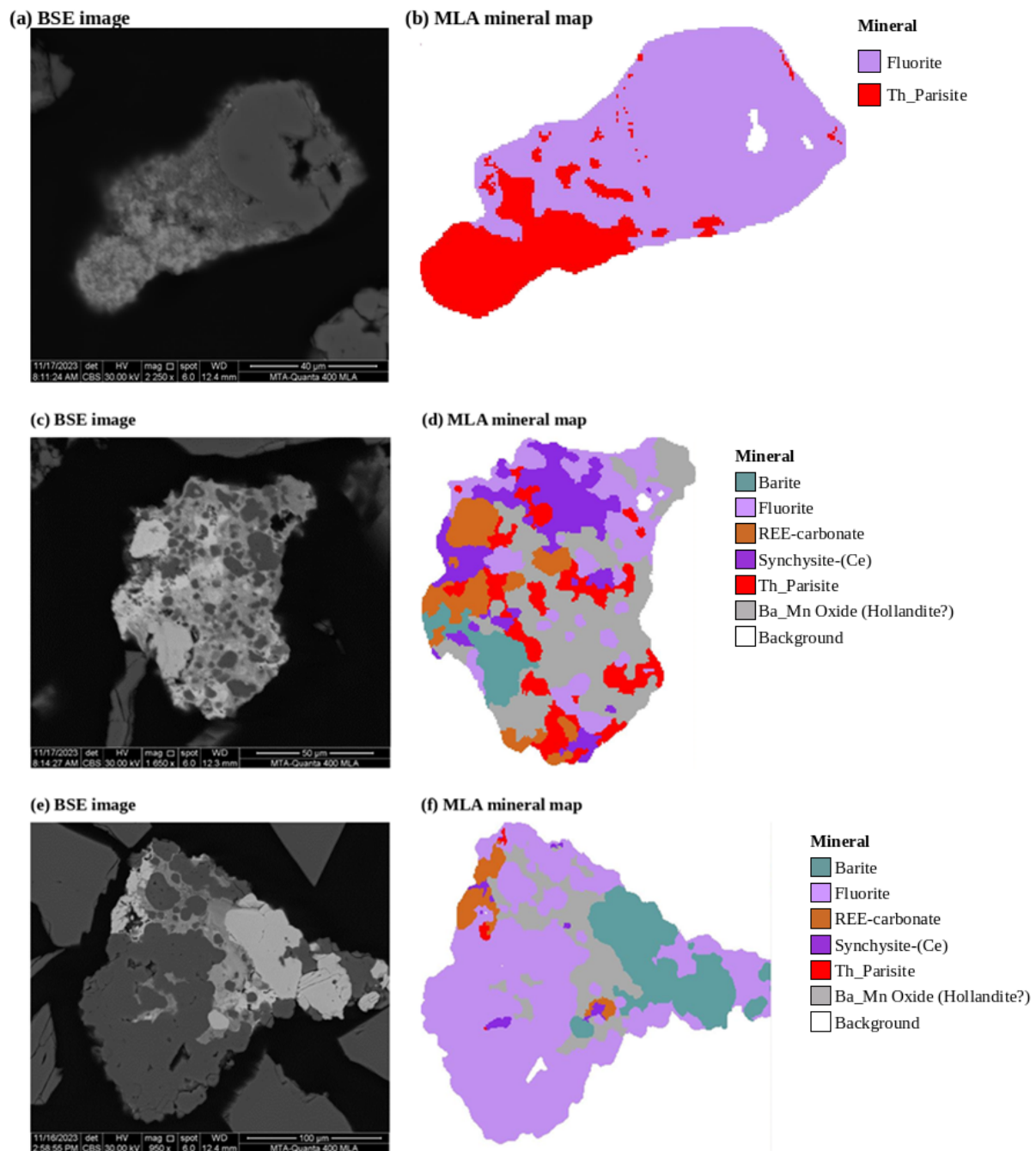


Fig. 7. Backscattered electron (BSE) images and corresponding MLA mineral maps of three pre-concentrate particles: (a, b) particle 1; (c, d) particle 2; (e, f) particle 3

3.3. Process design

Based on the experimental findings, a hydrometallurgical flowsheet is proposed for the Eskişehir-Beylikova pre-concentrate that achieves Th-U separation as an inherent outcome of the water leaching step itself, rather than as a downstream purification stage (Fig. 8). The pre-concentrate is first acid-baked (acid:sample 1:1, 250 °C, 60 min) to convert the Th, and U-bearing phases to their soluble sulfate forms. Notably, rejecting Th to the solid residue conventionally requires raising the acid baking temperature above 300 °C, at which Th converts into a thermodynamically stable thorium pyrophosphate-type phase (Demol et al., 2018; Zhu et al., 2015); however, this approach not only carries a substantial energy penalty but also locks Th into a highly refractory residue. In the present flowsheet, baking is instead kept at a moderate 250 °C, sufficient to convert Th to a soluble sulfate without forming this refractory phase, and the same selective Th rejection is achieved later, at the water leaching stage, simply through temperature control. The baked product is water-leached at 90 °C for 120 min, S/L 1:2 conditions selected specifically to exploit the divergent thermal response of Th and U established in Section 3.1: at 90 °C, the stability of Th-sulfate complexes is minimized, driving Th back out of solution and retaining it in the leach residue, while U, fully converted to highly soluble uranyl sulfate during baking, remains in solution irrespective of temperature. Filtration of the leached slurry, therefore, yields two products with sharply contrasting radio-element content: a Th-rich leach cake and a leach liquor containing essentially all dissolved U.

This outcome contrasts with the majority of Th-U separation routes reported in the literature, which rely on an additional dedicated step, selective precipitation, solvent extraction (SX), or ion exchange (IX) applied only after both elements have already been co-leached into solution (Zhu et al., 2015; Carrillo García et al., 2020; Kanojia et al., 2025). As outlined in the Introduction, these downstream routes carry substantial reagent, multi-stage circuit, and throughput costs. By achieving Th-U separation directly in the water-leaching step, the proposed flowsheet may reduce both the energy burden of high-temperature acid baking and these downstream separation costs, while still allowing U to be recovered as a discrete, concentrated stream in the leach liquor.

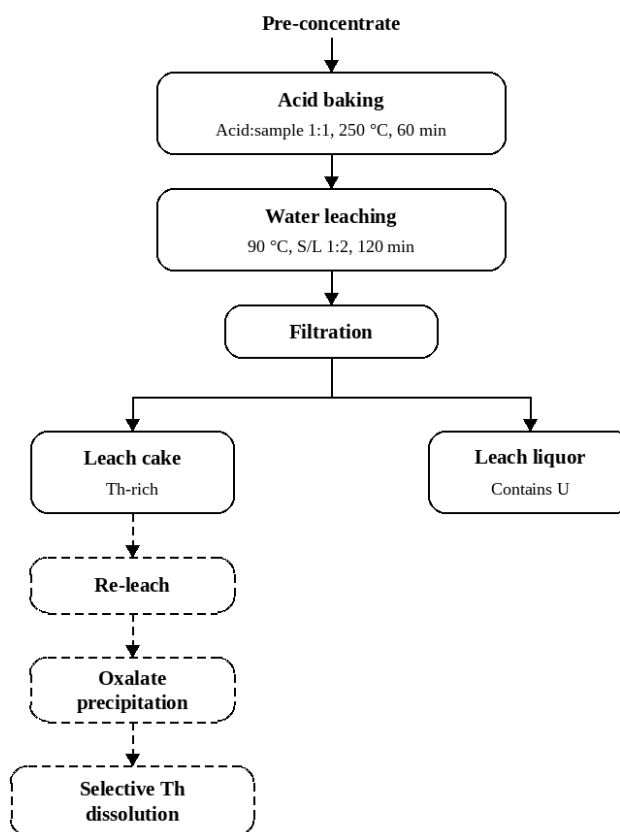


Fig. 8. Proposed hydrometallurgical flowsheet for the Eskişehir-Beylikova pre-concentrate via high-temperature and high solid ratio water leaching

In subsequent processing steps, the Th-rich leach cake, which also retains co-precipitated REEs, requires a dedicated re-leaching step (Fig. 8) to redissolve these species before further processing, since oxalate precipitation, the proposed next step, requires a solution feed rather than a solid cake. Following an approach analogous to that reported by Amer et al. (2013), Th can then be selectively separated via oxalate co-precipitation followed by selective re-dissolution of the Th oxalate. Consequently, rather than acting as a final purification step, this single-step leaching operation serves as a robust pre-treatment strategy that effectively isolates uranium and produces a Th-rich cake suitable for such downstream re-leaching and separation. REE recovery data under these conditions were not generated within the scope of the present study, which focuses specifically on Th-U partitioning during water leaching; the process claims herein are therefore restricted to this early Th-U separation step, and full REE recovery through the proposed downstream route remains to be experimentally verified.

4. Conclusions

This study demonstrates that thorium and uranium can be selectively separated directly within the water leaching step of an acid-baked REE pre-concentrate, without recourse to high-temperature baking, achieving an early separation between Th and U directly within the water leaching step. By systematically evaluating the effect of leaching temperature (25-90 °C) and time (15-120 min) on an acid-baked Eskişehir-Beylikova pre-concentrate under high pulp density conditions (S/L = 1:2), and by linking the observed dissolution behavior to the mineralogy of the Th- and U-bearing phases via MLA, this work establishes water leaching itself rather than precipitation, solvent extraction, or ion exchange as an effective, single-step Th-U separation mechanism, and translates this finding into a proposed hydrometallurgical flowsheet. The key findings of this study can be summarized as follows:

Th and U exhibit opposite responses to leaching temperature: Th recovery declines sharply at 70-90°C (falling to approximately 7%), while U recovery remains consistently high (~90%) and essentially independent of both temperature and time, yielding a recovery ratio (RR = U/Th) of approximately 13.3 at the optimum condition.

MLA confirms that Th and U are hosted exclusively by two distinct mineral phases Th-parisite (100% of Th) and plumbophyrochlore (100% of U), with Th-parisite occurring as poorly liberated, finely intergrown inclusions within the fluorite-, REE-carbonate-, and Ba-Mn-oxide-bearing matrix.

Based on these findings, a single-step flowsheet (acid baking at 250 °C followed by water leaching at 90 °C, S/L 1:2) is proposed that yields a Th-rich leach cake and a U-bearing leach liquor directly from filtration without requiring high-temperature baking, achieving this separation as an inherent outcome of the water leaching step itself, thereby also achieving an early separation between these two radioactive elements that simplifies downstream waste classification and handling.

Acknowledgments

A part of the results reported in this paper were presented at XX Balkan Mineral Processing Congress BMPC 2026 entitled "Evaluation of alternative sulfation agents for rare earth element pretreatment-a comparative study".

References

- AMER, T.E., ABDELLA, W.M., WAHAB, G.M.A., EL-SHEIKH, E.M., 2013. A suggested alternative procedure for processing of monazite mineral concentrate. *Int. J. Miner. Process.*, 125, 106–111.
- BAŞTÜRKÇÜ, E., 2026a. Evaluation of alternative sulfation agents for rare earth element pretreatment: a comparative study. In: Baştürkcü, E., Pural, Y.E., Üçerler Çamur, Z. (Eds.), 20th Balkan Mineral Processing Congress Proceedings Book. Turkish Mining Development Foundation, pp. 549–555.
- BAŞTÜRKÇÜ, E., ŞAVRAN, C., YÜCE, A.E., TİMUR, S.İ., 2022. Revealing the effects of mechanical attrition applied on Eskişehir-Beylikova REE ore utilizing MLA. *Miner. Eng.*, 186, 107733.
- BAŞTÜRKÇÜ, E., TİMUR, S.İ., YÜCE, A.E., 2026b. Leaching behaviour of rare earth elements from a complex ore via sulfuric acid baking: a geometallurgical perspective. *Can. Metall. Q.* <https://doi.org/10.1080/00084433.2025.2590802>
- CARRILLO GARCÍA, A., LATIFI, M., AMINI, A., CHAOUKI, J., 2020. Separation of radioactive elements from rare earth element-bearing minerals. *Metals*, 10(11), 1524.

- DEMOL, J., HO, E., SENANAYAKE, G., 2018. *Sulfuric acid baking and leaching of rare earth elements, thorium and phosphate from a monazite concentrate*. Hydrometallurgy, 179, 254–267.
- DU, C., LI, M., LIU, S., CHANG, H., HU, Y., 2010. *Present situation and suggestions for treatment of Baotou's rare earth concentrate*. Hydrometall. China, 29(1), 1–4.
- GUPTA, C.K., SINGH, H., 2003. *Uranium Resource Processing: Secondary Resources*. Springer-Verlag, Berlin, Germany.
- GÜRKAN, C., BAŞTÜRKÇÜ, E., TANÇ KAYA, B., OZDEMİR, O., YÜCE, A.E., 2026. *Depressant-free selective flotation of barite from a complex rare earth ore: A surface chemistry perspective*. JOM, 78(10), 9732–9743. <https://doi.org/10.1007/s11837-026-08644-4>.
- HUANG, X., LI, H., XUE, X., ZHANG, G., 2006. *Development status and research progress in rare earth hydrometallurgy in China*. J. Chin. Rare Earth Soc., 24(2), 130–133.
- KANAZAWA, Y., KAMITANI, M., 2005. *Rare earth minerals and resources in the world*. J. Alloy. Compd., 408–412, 1339–1343.
- KANOJIA, A., NYEMBWE, J.K., PETRANIKOVA, M., RETEGAN VOLLMER, T., EKBERG, C., 2025. *Solvent extraction and isolation strategies for uranium, thorium, and radium in rare earth element recovery from ores*. Solvent Extr. Ion Exch., 43 (5–6), 671–707.
- LAN, Q., ZHANG, X., NIU, F., LIU, D., SHEN, L., YANG, Y., 2025. *Recovery of rare earth elements and thorium from acid leaching residue of ionic rare earth concentrates*. J. Rare Earths, 43(4), 794–804.
- PAWLIK, C., 2013. *Recovery of rare earth elements from complex and low grade deposits*. ALTA 2013 Uranium-REE Conference, 25 May–1 June, Perth, Western Australia.
- PRATAMA, M.N., IQBAL, M., SANTOSO, I., ICHLAS, Z.T., WIYONO, 2025. *Selection of a pre-treatment method to extract and separate rare earth elements and thorium from Indonesian monazite*. IOP Conf. Ser.: Earth Environ. Sci., 1569(1), 012005.
- SATO, T., 1962. *The extraction of uranium (VI) from sulphuric acid solutions by di-(2-ethyl hexyl)-phosphoric acid*. J. Inorg. Nucl. Chem., 24, 699–706.
- SATO, T., 1968. *The extraction of thorium from hydrochloric acid solutions by di-(2-ethylhexyl)-phosphoric acid*. Z. Anorg. Allg. Chem., 358(19), 296–304.
- SATO, T., 1989. *Liquid-liquid extraction of rare-earth elements from aqueous acid solutions by acid organophosphorus compounds*. Hydrometallurgy, 22, 121–140.
- SOLDENHOFF, K., 2013. *Challenge in rare earth processing*. ALTA 2013 Uranium-REE Conference, 25 May–1 June, Perth, Western Australia.
- U.S. GEOLOGICAL SURVEY, 2025. *Mineral commodity summaries 2025*. U.S. Geological Survey. <https://doi.org/10.3133/mcs2025>
- VERCOUTER, T., VITRORGE, P., AMEKRAZ, B., MOULIN, C., 2008. *Stoichiometries and thermodynamic stabilities for aqueous sulfate complexes of U(VI)*. Inorg. Chem., 47 (6), 2180–2189.
- WANG, J., WANG, X., 2006. *Comprehensive exploitation and cleaning production technology of Bayan Obo rare earth ore*. Chin. Rare Earth, 27(1), 103–105.
- ZHU, Z., PRANOLO, Y., CHENG, C.Y., 2015. *Separation of uranium and thorium from rare earths for rare earth production – A review*. Miner. Eng., 77, 185–196.
- ZIELEN, A.J., 1959. *Thermodynamics of the sulfate complexes of thorium*. J. Am. Chem. Soc., 81(20), 5022–5028.