Hydrochloric Acid Leaching Approaches for Tantalum and Niobium Pentoxides Enrichment in 1st Tin Slag from Bangka Island

Sulaksana Permana12*, Rafdi Abdul Majid3, and Johny Wahyuadi Soedarsono1
1Centre of Mineral Processing and Corrosion Research, Department of Metallurgy and Materials, Universitas Indonesia, 16424 Depok, Jawa Barat, Indonesia
2Institut Sains dan Teknologi Al-Kamal Jl. Raya Al Kamal No.2 Kedoya Selatan, Kebon Jeruk Jakarta Barat, DKI Jakarta 11520, Indonesia
3Department of Chemical Engineering, Faculty of Industrial Technology, Universitas Muslim Indonesia, Panakukang Makassar 90231

*sulaksana@yahoo.com

Abstract. Indonesia is the second largest tin producer in the world that resulting abound of by-product. This by-product contains some of valuable metal oxides, such as 0.30% tantalum (Ta2O5) and 0.58% niobium pentoxide (Nb2O5). The supply deficit of these minerals induces researchers worldwide to conduct investigations. This study explored the effect of roasting at 900°C, NaOH quenching, and HCl leaching with 45, 65, and 85°C temperature variables and 4M, 6M, 8M concentration variables. These measures were employed to upgrade the contents of tantalum and niobium pentoxide (TNP). The filtration products of quenching and leaching were characterized by X-Ray Fluorescence (XRF). On the one hand, 900°C roasting and NaOH quenching decreased the 100 mesh-measuring TNP contents, 0.14% Ta2O5 and 0.2% Nb2O5. On the other hand, HCl leaching with a 65°C temperature variable showed an optimal yield ratio of 3.75 Ta2O5 and 3.57 Nb2O5. The total yield ratio of TNP is 3.63.

Keywords: 1st tin slag, tantalum and niobium pentoxide, roasting, quenching, hydrochloric acid leaching

1. Introduction
Tantalum and niobium are elements which are not found freely in nature, but are always associated with several types of metals to form oxide minerals. Tantalum and niobium are included in the category of critical metal in the world, this is related to the assessment results of 78 types of metal produced by the European Commissioning in 2017 [1]. On the other hand, both of two metals have many uses in high-tech applications such as the aircraft industry, electronics, super alloys, catalysts, capacitors, energy and superconductors [2]. A survey on tantalum supply and demand in 2013 indicated that in 2015, this chemical element would undergo a deficit of 79,000 lbs. and in 2016, of 212,000 lbs. Roskill estimated that the total demand for tantalum would have been growing by 3.3% CAGR for the next nine years, 2017-2026 [3]. Research on a tantalum secondary source from tin slag has been conducted for several decades, between 1964 and now [4-24].
The general process of tantalum extraction involves smelting raw materials comprised of tin concentrates, flux additives, and reducing agent [24]. This process manufactures a by-product called 1st tin slag (1TS) which holds approximately 20 % tin. The next process is a tin recovery which also employs smelting. This process then produces 2nd tin slag (also called as Bangka Tin slag (BTS) [23]) by-product.

After 2011, research on tantalum from tin slag relatively increased. The rise of exploration was driven by a tantalum deficit. Supply of tantalum is predicted to exist until 2020 [25]. Studies of metal recovery mostly involved roasting and quenching since these two processes could create more pores and extend the wet touched areas. These wet touched areas would improve the effectiveness of dissolution or leaching. Roasting and quenching were applied to research on rich husk [26] and tin slag [23] to develop the quality of the substance which would be recovered.

Both dissolution and leaching are some of the methods used for metal extraction. In this investigation, leaching was employed to dissolve oxides. Several prior explorations also exploited this method for processing REE and scandium elements [27-31]. A phase diagram thermodynamic analysis of Ta2O5, Nb2O5 and rare earth was done using HSC Chemistry [10], [32]. This device was also involved in research focusing on (1) lithium extraction from lepidolite [33-35], (2) roasting of nickeliferous and smelting of ferronickel, both of which were extracted from laterite [36,37], and (5) simulation of cesium reaction during a high temperature voloxidation [38]. In additional, the use of HSC Chemistry was for training simulator for processing minerals [39] and conversion of alternative hydrocarbon-based feedstock to hydrogen [40].

Some previous studies shows that the use of alkaline and acid solvents can increase contents of tantalum and niobium oxide in tin slag through the dissolution of other oxides such as Fe2O3 and SiO2 [41], this is also explained by several other studies [42-45,46]. In addition, according to A. Suharyanto, the use of sodium hydroxides (NaOH) can dissolve amphoteric compounds in major elements such as SnO2, SiO2, Al2O3 and Zr [47], this is related with some research [48-51].

This study aims to determine the effect of heat treatment and alkaline or acid leaching on Bangka tin slag (BTS) to increasing the contents of tantalum and niobium oxide through roasting followed by quenching with NaOH, leaching with HCl as base solvents in several temperature and concentration variables.

2. Experimental method

2.1. Materials & Apparatus
In this investigation, tin slag was taken from smelters in Bangka and Belitung Islands, Indonesia. The characterization using XRF Spectro Xepos Ametek represented above 20% SnO2. This indicated that the tin slag was 1TS. This process employed (1) a Pyrex beaker glass 1000 ml IWAKI TE-32 Asahi Techno Glass, (2) a measuring cup 100 ml ± 0.1mlIWAKI CTE33 Asahi Glass, (3) a Pyrex measuring pumpkin 1000 ml ± 0.15 mlIWAKI TE-32 Asahi Techno Glass, (4) a stirring bar, (5) aquades, (6) a magnetic stirrer IKA® C-MAG HS 7, and (7) a magnetic stirrer Yellow Line MSH Basic IKA.

2.2. Methods
First, 1TS was characterized through XRF to find its chemical compounds. These samples were, then, roasted at 900°C for 2 hours, quenched by NaOH 0.5 M, dewatered, and sieved with +100, -100+120, -120+180 mesh. The +100 mesh products of quenching were divided into two groups. Some would undergo HCl leaching with temperature variables and others with concentration variables. The first process exploited 6M HCl at 25°, 45°, and 65°C while the second one used 4M, 6M, and 8M HCl. The detail procedures can be seen in Figure 1.
2.3. Abbreviations and Terminology
To easily follow the discussion, several abbreviations are provided. 1TS is an abbreviation for 1st tin slag while 1TS-RQS means first tin slag which has been roasted at 900°C and NaOH quenched.

Other abbreviations are TNP, MOO, and EMO. Tantalum and niobium pentoxide is abbreviated as TNP while MOO is a short-term for major other oxides. These oxides contain high concentration of SnO$_2$, Fe$_2$O$_3$, SiO$_2$, K$_2$O, Al$_2$O$_3$, Na$_2$O, TiO$_2$, and ZrO$_2$. Lastly, other elements or oxides that do not belong to both TNP and MOO are summed up as elements or minor other oxides (EMO).

2.4. Characterization and Chemical Software
X-ray fluorescent (XRF) was performed to do the characterization of sample 1TS, 1TS-RQS, ITS-RQS leached. X-ray diffraction (XRD) was performed to do the characterization of sample 1TS. Additionally, thermodynamic software was included to enrich discussion materials [52].

3. Results and Discussion
The research results are comprised of a variety of data which have been characterized by XRF. They are 1TS, 1TS-RQS, ITS-RQS leached with temperature variables, and ITS-RQS leached with concentration variables.

3.1 Composition of 1st tin slag
Table 1 illustrates the contents of Ta$_2$O$_5$ and Nb$_2$O$_5$ which are 0.29(7) % and 0.58(2) % respectively, and there is CaO that is less than 3 digits. Compared to BTS [23], this 1TS holds several oxides such as SiO$_2$, Al$_2$O$_3$, TiO$_2$, ZrO$_2$, and CaO which are less than BTS contents, and Fe$_2$O$_3$ and SnO$_2$ which are more than BTS contents.

|        | Ta$_2$O$_5$ (%) | Nb$_2$O$_5$ (%) | Fe$_2$O$_3$ (%) | SiO$_2$ (%) | Al$_2$O$_3$ (%) | TiO$_2$ (%) | ZrO$_2$ (%) | CaO (%) | SnO$_2$ (%) |
|--------|-----------------|-----------------|-----------------|-------------|-----------------|-------------|-------------|---------|-------------|
|        | 0.29            | 0.58            | 14.72           | 11.34       | 4.48            | 4.92        | 2.41        | <0.00   | 22.95       |
ITS characterization with X-ray diffraction (XRD) is shown in Figure 2. XRD analysis result shows that tantalum was compounding with silica and titanium; Si$_{0.72}$Ta$_{3.28}$ peaks are seen at 2θ: 34.2°, 60.9° and 72.6°, while Ta$_{0.15}$Ti$_{0.85}$ peaks are at 2θ: 34.7°, 60.9° and 79.9°. But niobium, it was compounding with silica, and this is seen in Nb$_5$Si$_3$ with its peaks at 2θ: 31.3°, 50.0° and 90.3°. XRD analysis results show MOO elements in the form of AlF$_3$ where the peaks are seen at 2θ: 30.8°, 52.4° and 54.8°.

![Figure 2. X-Ray Diffraction analysis of the 1st Bangka tin slag (1TS).](image)

3.2 Roasting and sodium hydroxide quenching

In comparison with 1TS contents, those of 1TS-RQS decreased by about two-thirds. Ta$_2$O$_5$ and Nb$_2$O$_5$ the size of 100 mesh became 0.10(4) % and 0.20(0) %, the size of -100+120 mesh 0.12(6) % and 0.18(6) %, and the size of -120+180 mesh 0.14(7) % and 0.20(6) % respectively. Figure 3 (a) shows a decline in the contents of Ta$_2$O$_5$ and Nb$_2$O$_5$ after being roasted at 900°C and NaOH quenched.

![Figure 3. Roasted and NaOH quenched products after being sieved: (a) the contents of TNP (b) the contents of MOO.](image)

In Figure 3 (b), the contents of all MOO and SnO$_2$ sizes declined. In this case, those measuring 100 mesh became dominant. Among all, the 100 mesh TiO$_2$ underwent the highest solution, from 4.91(6) %
in 1TS to 1.48(2) % in 1TS-RQS. In other words, it only left 30 %. Other highly soluble oxides whose concentration declined were SnO$_2$, SiO$_2$, and ZrO$_2$. The three substances left almost merely 40%.

Ferri and silica oxides become the majority of 1TS elements. This indicates that this substance is categorized as a ceramic compound. The structure of ceramic is so brittle that it can be broken before deformation occurs. This is driven by the structure and development of a crack over the materials [53]. Tin slag composition in Table 1 shows 22.95 % SnO$_2$ which indicates that the tin slag is 1TS. Roasting of tin slag brings about reactions which go from the surface to the core of particles. These reactions are controlled by oxygen diffusion that comes in and sulfur dioxide that comes out through oxide layers. Roasting would develop porosity, a thermal expansion which creates a micro crack [54], and extend the surface of solids. This would encourage the optimization of leaching [55].

Roasting and sodium hydroxide quenching dissolved near two-thirds of TNP and MOO contents. HSC Chemistry simulation, as in Table 2, represents a reaction of $\Delta G$ negative to Ta$_2$O$_5$, SiO$_2$, TiO$_2$, and ZrO$_2$ with NaOH.

**Table 2.** Values of $\Delta G$ negative at 900°C and NaOH solution.

| No | Reaction Equation | $\Delta G_{900}$ (kcal) |
|----|-------------------|-------------------------|
| 1  | Ta$_2$O$_5$ + NaOH = NaTaO$_3$ + H$_2$O | -0.322 |
| 2  | SiO$_2$ + NaOH = Na$_2$SiO$_3$ + H$_2$O | -16.881 |
| 3  | TiO$_2$ + NaOH = Na$_2$Ti$_2$O$_5$ + H$_2$O | -23.040 |
| 4  | ZrO$_2$ + NaOH = Na$_2$ZrO$_3$ + H$_2$O | -2.936 |

Correlation between Table 2 and the result of HSC Chemistry simulation represents that $\Delta G$ negative values produce compounds in a Pourbaix diagram on pH 13.69 (8), it’s means that the reaction it will be process when contact in that condition. Figure 4 shows the Pourbaix diagram of Ta$_2$O$_5$, SiO$_2$, TiO$_2$, and ZrO$_2$ with NaOH.

**Figure 4.** Pourbaix diagram of Ta$_2$O$_5$, SiO$_2$, TiO$_2$, and ZrO$_2$ with NaOH (HSC Chemistry 6).
3.3 Hydroxide chloride leaching

3.3.1 Leaching with temperature variables. Leaching with 45°C and 65°C temperature variables represented a significant upgrade in both TNP and MOO contents. The contents of TNP correlated with the rise of temperature. Meanwhile, leaching at 65°C resulted in the maximum contents. Figure 5 (a) shows the upgrade in TNP contents referred to (1) TNP from the 100 mesh ITS-RQS and (2) TNP from the 25°C, 45°C, and 65°C leaching.

![Graph showing TNP content with temperature variables.](image)

Figure 5. Leached products with temperature variables: (a) the contents of TNP (b) the contents of MOO.

Figure 5 (b) shows an increase in MOO contents leached at 25°C, 45°C, and 65°C. Among the three temperatures, 45°C and 65°C dominated the rise. The related substances were SnO₂, SiO₂, and ZrO₂.

Table 3. Values of ΔG negative of ITS-RQS solution of HCl.

| Reaction Equation | ΔG<sub>25</sub> (kcal) | ΔG<sub>45</sub> (kcal) | ΔG<sub>65</sub> (kcal) |
|------------------|---------------------|---------------------|---------------------|
| NaTaO₃ + HCl(g) → Ta₂O₅+NaCl+H₂O | -24.683 | -23.604 | -22.603 |
| Na₂SiO₃ + HCl(g) → SiO₂+NaCl+H₂O | -48.690 | -47.589 | -46.503 |

In Table 3 and Figure 6, a thermodynamic reaction occurs. Ta₂O₅ was found in pH -0.77(8) in which 6M HCl was employed. In other words, NaOH quenched tantalum pentoxide, if HCl leached, will produce the same substance, Ta₂O₅. Dissolution at all temperature variables does not influence the tantalum compounds. However, Table 4 shows that the yield ratio shows 3.75 as the maximum value of Ta₂O₅ at 65°C.

Table 4. Yield ratio of leaching with temperature variables.

| Initial +100 # | Content 25°C | Yield Ratio 25°C | Content 45°C | Yield Ratio 45°C | Content 65°C | Yield Ratio 65°C |
|---------------|--------------|-----------------|--------------|-----------------|--------------|-----------------|
| Ta₂O₅         | 0.10(4)      | 0.13(8)         | 1.32         | 0.37(4)         | 0.39(0)      | 3.75            |
| Nb₂O₅         | 0.20(1)      | 0.21(5)         | 1.07         | 0.67(9)         | 0.71(5)      | 3.57            |
| Total         | 0.30(5)      | 0.35(3)         | 1.16         | 1.05(3)         | 1.10(6)      | 3.63            |

Yield Ratio = content before process/content after process.
3.3.2 Leaching with concentration variables. HCl leaching with concentration variables did not show a significant upgrade in the contents of TNP. However, MOO contents including SiO$_2$, Al$_2$O$_3$, and TiO$_2$ dominantly rose after being HCl leached with concentration of 4M. The increase in concentration did not correlate with the contents of MOO. Figure 7 (a) shows an upgrade in TNP contents referred to (1) TNP contents from 1TS-RQS measuring 100 mesh and (2) TNP contents leached with concentration of 4M, 6M, and 8M. Figure 7 (b) shows an increase in MOO contents leached at 4, 6, and 8M.

HCl leaching thermodynamic reactions with concentration of 4, 6 and 8 M (pH -0.602, -0.778 and -0.9030) at 25°C on tantalum oxides and silica can be seen in Figure 8. The extension of pH from -
0.602 to -0.9030 does not indicate any compound reactions. In Table 5, no concentration variables show a significant rise in the yield ratio.

**Table 5.** Yield ratio of leaching with concentration variables.

|       | Initial Content | Yield Ratio | Content | Yield Ratio | Content | Yield Ratio |
|-------|-----------------|-------------|---------|-------------|---------|-------------|
| +100 # | Ta₂O₅            | 0.10(4)     | 1.74    | 0.14(8)     | 1.42    | 0.18(3)     |
|       | Nb₂O₅            | 0.20(1)     | 1.28    | 0.23(7)     | 1.18    | 0.25(9)     |
|       | Total            | 0.30(5)     | 1.44    | 0.38(6)     | 1.27    | 0.44(3)     |

Yield Ratio = content before process/content after process.

4. Conclusion
The main results of thermal treatment (roasting) followed by sodium hydroxide quenching and acid leaching by HCl obtained in this study can be summarized that roasting of tin slag in 900°C and NaOH quenching make TNP and MOO contents within residues dissolve in filtrates. While, leaching process of 1TS-RQS with HCl are increase in each temperature variables, but the maximum contents of TNP at 65°C with yield ratio of 3.75 Ta₂O₅ and 3.57 Nb₂O₅. It’s also seen in the thermodynamic analysis with Gibbs Energy and Pourbaix Diagram are negative. But leaching process of 1TS-RQS with HCl in some concentration variables does not indicate any dominance in yield ratio.

Acknowledgment
Special thanks to Research Centre for Metallurgy, Indonesian Institute of Sciences-LIPI, Indonesia and Mechanical Engineering Department, Politeknik Negeri Jakarta, Depok 16425, Indonesia.

References
[1] European Commissioning, “Report on Critical Raw Materials for the EU- critical raw materials profiles,” 2015.
[2] M. S. P. REFRAAM, “Niobium and tantalum,” the European Union’s Horizon 2020 research and innovation programme under grant agreement No 688993
[3] Roskill Information Services Ltd., Tantalum: Global Industry, Markets and Outlook to 2020, 12th ed. London: Roskill Information Services, 2016.
[4] Sidney M. Heins, “US Patent 3.129.896-Separation of Heavy Metal Contituents of Tin Slag,” 3.129.896, 1964.
[5] J. A. Cenerazzo, C. E. Mosheim, Zionsville, and Corrodo E. Marvisa, “US Patent 3,585,024-Up Grading The Tantalum and Columbium Contents of Tin Slag,” 1971.
[6] I. Gaballah, E. Allain, M.-C. Meyer-Joly, and K. Malal, “A Possible Method for the Characterization of Amorphous Slags Recovery of Refractory Metal Oxides from Tin Slags,” Metall. Trans., vol. 23, no. June, pp. 249–259, 1992.
[7] A. K. Suri, C. Subramanian, and C. K. Gupta, “Investigations on flowsheet development for the treatment of niobium-tantalum bearing cassiterite,” Miner. Process. Extr. Metall. Rev., vol. 9, no. 1–4, pp. 293–303, 1992.
[8] M. Y. Sulaiman, “Simultaneous determination of thorium and uranium in tin slag,” Sci. Total Environ., vol. 131, pp. 187–195, 1993.
[9] I. Gaballah and E. Allain, “Recycling of Strategic Metals from Industrial Slag by a Hydro-and Pyrometallurgical Process,” Resour. Conserv. Recycl., vol. 10, no. 1–2, pp. 75–85, Apr. 1994.
[10] E. Allain, M. Djona, and I. Gaballah, “Kinetics of Chlorination and Carbochlorination of Pure Tantalum and Niobium Pentoxides,” Metall. Mater. Trans. B, vol. 28, no. April, pp. 223–233, 1997.
[11] I. Gaballah, E. Allain, and M. Djona, “Extraction of Tantalum and Niobium from Tin Slags by Chlorination and Carbochlorination,” Metall. Mater. Trans., vol. 28, no. June, 1997.
[12] C. Subramanian, A. K. Suri, and B. Atomic, “Recovery of Niobium and Tantalum from Low Grade Tin Slag - A Hydrometallurgical Approach,” Environ. Waste Manag., pp. 100–107, 1998.
[13] P. K. Jena and E. A. Brocchi, “Studies on the Kinetics of Carbon Tetrachloride Chlorination of Refractory Metal Oxides,” Min. Proc. Ext. Met. Rev., vol. 22, pp. 597–613, 2001.
[14] N. T. Mudzanapabwe, O. S. Chinyamakobvu, and D. J. Simbi, “In Situ Carbothermic Reduction of a Ferro-Columbite Concentrate in The Recovery of Nb and Ta as Metal Matrix from Tin Smelting Slag Waste Dump,” Mater. Des., vol. 25, pp. 297–302, 2004.
[15] Byung-Su Kim, And, and Y.-Y. Choi, “Kinetics of the Chlorination Reaction of Tantalum Pentoxide with Carbon Tetrachloride Gas,” Mater. Trans., vol. 46, no. 9, pp. 2102–2106, 2005.
[16] MS, M. Yusoff, and M. Muslimin, “Uranium and Thorium Analysis In Tin Slag by EDXRF,” J. Nucl. Relat. Technol., vol. 4, pp. 177–180, 2007.
[17] E. A. Brocchi and F. J. Moura, “Chlorination Methods Applied to Recover Refractory Metals from Tin Slags,” Miner. Eng., vol. 21, pp. 150–156, 2008.
[18] M. Meor Sulaiman, M. Y, Muslimin, “Quantitative Analysis of Uranium and Thorium in Local Zircon and Tin Slag by The EDXRF Technique,” J. Nucl. Relat. Technol., vol. 7, no. 1, pp. 21–28, 2010.
[19] C. Francois, “Patent_WO_2013_040694_A1_3.pdf,” 2013.
[20] J. U. Odo, W. C. Okafor, S. O. Ekpe, And, and C. C. Nwogbu, “Extraction of Niobium from Tin Slag,” Int. J. Sci. Res. Publ., vol. 4, no. 11, pp. 1–7, 2014.
[21] S. I. Angadi, T. Sreenivas, H.-S. Jeon, S.-H. Baek, and B. K. Mishra, “A Review of Cassiterite Beneficiation Fundamentals and Plant Practices,” Miner. Eng., vol. 70, pp. 178–200, Jan. 2015.
[22] M. Y. Meor Sulaiman, N. S. Kamaruzaman, and A. K. Zahari, “Standardless EDXRF Analysis Methods of U and Th in Malaysian Tin Slag Waste,” Mater. Sci. Forum, vol. 840, pp. 410–415, 2016.
[23] S. Permana, J. W. Soedarsono, A. Rustandi, and et.al, “Other Oxides Pre-removed from Bangka Tin Slag to Produce a High Grade Tantalum and Niobium Oxides Concentrate,” in IOP Conference Series : Materials Science and Engineering, 2016, vol. 012006, p. 10.
[24] R. Smith, “An Analysis of the Processes for Smelting Tin,” Bull. Peak Dist. Mines Hist. Soc., vol. 13, no. 2, p. 9, 1996.
[25] R. U. A. Laura Talens Peiro, Gara Villalba Mendez, “Rare and Critical Metals as By-products and The Implications for Future Supply,” 2011.
[26] A. Maksum, A. Rustandi, S. Permana, and J. W. Soedarsono, “Influence of Roasting-Quenching Pretreatment on the Rice Husk Silica Prepared by Calcination Method,” vol. 020009, 2017.
[27] J. Soedarsono, M. Burgard, Z. Asfari, and J. Vicens, “Liquid-liquid extraction properties of
25,27-dicarboxy-26,28-dimethoxy-5,11,17,23-tetra-tert- butylcalix [4] arene towards the alkali ions,” *Solvent Extr. Ions Exch.*, vol. 13, no. 4, pp. 755–769, 1995.

[28] J. Soedarsono, M. Burgard, Z. Asfari, and J. Vicens, “Liquid-liquid Extraction of Rare Earth (III) Ions by 25,27-dicarboxy-26,28-dimethoxy-5,11,17,23- tetra- tert-butylcalix[4] arene,” in *XVII International Symposium on Macrocyclic Chemistry*, 1993.

[29] J. Soedarsono, A. Hagege, M. Burgard, Z. Asfari, and J. Vicens, “Liquid-liquid Extraction of Rare Earth Metals Using 25,27-dicarboxy-26,28-dimethoxy- 5,11,17,23-tetra- tert-butylcalix[4] arene,” *J. Phys. Chem.*, vol. 100, no. 4, pp. 477–481, 1996.

[30] Y. Sohama, H. Isaka, and H. Watanabe, “Patent WO2002022895 Method for producing metal or metal compound comprising process of treating with fluorine and adjusted raw material used therein,” WO2002022895, 2003.

[31] W. Wang, Y. Pranolo, and C. Y. Cheng, “Recovery of Scandium from Synthetic Red Mud Leach Solutions by Solvent Extraction with D2EHPA,” *Sep. Purif. Technol.*, vol. 108, pp. 96–102, 2013.

[32] S. Permana, A. Rustandi, and R. A. Majid, “Thermodynamic Analysis With Software : A Case Study of Ugrading Rare Earth Elements Content in Bangka Tin Slag,” *Far East J. Electron. Commun.*, vol. 17, no. 5, pp. 1211–1220, 2017.

[33] V. T. Luong, D. J. Kang, J. W. An, M. J. Kim, and T. Tran, “Factors affecting the extraction of lithium from lepidolite,” *Hydrometallurgy*, vol. 134–135, pp. 54–61, 2013.

[34] V. T. Luong, D. J. Kang, J. W. An, D. A. Dao, M. J. Kim, and T. Tran, “Iron sulphate roasting for extraction of lithium from lepidolite,” *Hydrometallurgy*, vol. 141, pp. 8–16, 2014.

[35] T. T. Hien-Dinh, V. T. Luong, R. Gieré, and T. Tran, “Extraction of lithium from lepidolite via iron sulphide roasting and water leaching,” *Hydrometallurgy*, vol. 153, pp. 154–159, 2015.

[36] C. A. Pickles, J. Forster, and R. Elliott, “Thermodynamic analysis of the carbothermic reduction roasting of a nickelferous limonitic laterite ore,” *Miner. Eng.*, vol. 65, pp. 33–40, 2014.

[37] D. R. Swinbourne, “Understanding ferronickel smelting from laterites through computational thermodynamics modelling,” *Miner. Process. Extr. Metall.*, vol. 123, no. 3, pp. 127–140, 2014.

[38] M. K. Jeon, J. M. Shin, J. J. Park, and G. Il Park, “Simulation of Cs behavior during the high temperature voloxidation process using the HSC chemistry code,” *J. Nucl. Mater.*, vol. 430, no. 1–3, pp. 37–43, 2012.

[39] T. Roine, J. Kaartinen, and P. Lamberg, “Training simulator for flotation process operators,” in *IFAC Proceedings Volumes (IFAC-PapersOnline)*, 2011, vol. 18, no. PART 1, pp. 12138–12143.

[40] E. Turpeinen, R. Raudaskoski, E. Pongrácz, and R. L. Keiski, “Thermodynamic analysis of conversion of alternative hydrocarbon-based feedstocks to hydrogen,” *Int. J. Hydrogen Energy*, vol. 33, no. 22, pp. 6635–6643, 2008.

[41] H. F. Putra and N. S. Aryanti, “Upgrading tantalum and niobium oxides content in Bangka tin slag with double leaching Upgrading tantalum and niobium oxides content in Bangka tin slag with double leaching,” 2018.

[42] R. A. Majid, A. Rustandi, and S. Permana, “Simulation of Tantalum and Niobium Pentoxides Extraction from Bangka Tin Slag Waste,” vol. 24, no. 1, pp. 767–772, 2018.

[43] E. Allain, N. Kanari, F. Diot, and J. Yvon, “Development of a process for the concentration of the strategic tantalum and niobium oxides from tin slags,” *Miner. Eng.*, vol. 134, no. January, pp. 97–103, 2019.

[44] J. W. Soedarsono and S. Permana, “Other Oxides Pre-removed from Bangka Tin Slag To Produce a High Grade Tantalum and Niobium Oxides Concentrate Other” IOP Conf. Ser.: Mater. Sci. Eng. 131 012006, 2016.

[45] W. Kartika, R. A. Majid, and D. Navanti, “Studi Pemanfataan Limbah Terak Timah 2 Bangka Sebagai Sumber Sekunder Unsur Unsur Skandium,” vol. 19, no. 1, pp. 8–17, 2019.

[46] S. Permana, A. Maksum, and J. Wahyuadi, “Enrichment On Bangka Tin Slag ’ S Tantalum And Niobium Oxide Contents Through Non-Fluoride Process,” January 2018.
[47] A. Suharyanto, E. Sulistiyono, and F. Firdiyono, ‘Pelarutan Terak Timah Bangka Menggunakan Larutan NaOH,’ vol. 15314, 2014.,” vol. 15314, p. 15314, 2014.

[48] R A Majid, S. Permana and J W Soedarsono, “Peningkatan Kadar Tantalum & Niobium Oksida dari Terak Timah Bangka Menggunakan Pilarut NaOH dilanjutkan dengan HNO3 dan H3PO4,” Journal of Chemical Process Engineering Vol. 4, No.01, 2019.

[49] S. F. Vincia, S. Permana, A. Maksum, and J. W. Soedarsono, “Study to Enhance Tantalum and Niobium Contents in Bangka Tin Slag by NaOH and HClO 4 Leaching” IOP Conf. Ser.: Mater. Sci. Eng. 547 012049, 2019.

[50] S. Permana, A. Maksum, and J. W. Soedarsono, “Enrichment of Tantalum and Niobium Contents in Bangka Tin Slag by NaOH and H 3 PO 4 Leaching” IOP Conf. Ser.: Mater. Sci. Eng. 547 012050, 2019.

[51] P. Harcourt, E. S. Science, V. Schools, and M. Board, “Extraction of Niobium from Tin Slag,” vol. 4, no. 11, pp. 1–7, 2014.

[52] A. Roine, T. Kotiranta, and et.al, “HSC Chemistry, Version 6.0,” vol. 03, no. 5641. Outokumpu Research Oy, p. 5641, 2006.

[53] P. W. R. Beaumont, “On the problems of cracking and the question of structural integrity of engineering composite materials,” Appl. Compos. Mater., vol. 21, no. 1, pp. 5–43, 2014.

[54] J. Chang, J. Peng, L. Zhang, and J. Cheng, “Extraction of Indium from Zinc Oxide Flue Dust by Microwave Sulfation Roasting and Water Leaching,” Drying, Roasting, Calcining Miner., pp. 35–42, 2015.

[55] E. O. Olanipekun, “Kinetics of leaching laterite,” Int. J. Miner. Process., vol. 60, no. 1, pp. 9–14, 2000.