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# Strategies for the Recovery of Tungsten from Wolframite, Scheelite, or Wolframite–Scheelite Mixed Concentrates of Spanish Origin

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**Abstract**

Among the strategic materials considered by the EU, tungsten is included; thus, investigations about the recovery of this metal both from natural and recyclable sources are of interest. In this work, we presented an investigation about the recovery of tungsten based on the treatment of three tungsten-bearing concentrates: scheelite (29% W), wolframite (50% W), and mixed scheelite–wolframite (29% W). All of these come from a cassiterite ore of Spanish origin. The characteristics of each concentrate pave the procedure to be followed in each case. In the case of the wolframite concentrate, the best results were derived from the leaching of the ore with NaOH solutions, whereas the treatment of the scheelite concentrate benefits from an acidic (HCl) leaching. The attack of the mixed concentrate is only possible by a previous roasting step (sodium carbonate and 700–800 °C) followed by a leaching step with water. In the acidic leaching, tungstic acid ($H_2WO_4$) was obtained, and the alkaline–water leaching produces $Na_2WO_4$ solutions from which pure synthesized scheelite is precipitated.

**Keywords:** tungsten; recovery; recycle; leaching; scheelite; wolframite

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## 1. Introduction

Tungsten is another metal included in the 2023 European Union list of Critical and Strategic Raw Materials (CRM and SRM, respectively). China is the largest producer (86%) of tungsten in the world and the largest supplier (32%) of this metal to the EU [1]. The strategic importance of this element dates back to the era of the First and Second World Wars, where it was used in tungsten alloy radiation shielding (pure tungsten), military and aerospace applications (tungsten alloys), welding tungsten alloys and jewelry (tungsten carbide), fluorescent lighting (Ca and Mg tungstates), etc.

The main sources for tungsten production are its raw materials, scheelite ($CaWO_4$) and wolframite ($(Fe,Mn)WO_4$), though its recovery is also associated with the presence of these mineral species in cassiterite ores; thus, tungsten is sometimes a subproduct of the tin industry. Wolframite, together with cassiterite, coltan, and gold, is known as a conflict mineral or 3TG (tin, tantalum, tungsten, gold).

Both pyrometallurgical and/or hydrometallurgical processing have been widely considered for the recovery of this element from different sources. Among recent publications, different flowsheets in the treatment of scheelite, wolframite, and secondary sources have

been reviewed [2]. A description of leaching procedures on scheelite [3] or the use of hydrometallurgical processing in the recovery of tungsten from raw or secondary materials [4] can be found.

In the recovery of tungsten from secondary sources, the treatment of spent selective catalytic reduction denitration catalysts by $Na_2S$ alkali leaching and calcium precipitation [5], the recovery of this strategic element from tungsten fine mud [6], or the recycling of tungsten-filled vinyl–methyl–silicone-based flexible shielding materials via pyrolysis and ultrasonic cleaning procedure [7] have been considered.

Tungsten scraps were utilized in the production of peroxotungstic acid or PTA. In this investigation, the scraps were subjected to the following steps: preliminary oxidation of waste tungsten carbide, reduction in intermediate-valence tungsten oxides, and subsequent dissolution in hydrogen peroxide to produce this peroxotungstic acid [8].

Waste fluidized catalytic cracking (FCC) catalysts were used to recover Mo and W using an oxidation treatment–alkali roasting–wet mill leaching operational step [9].

Real W-Ni electroplating wastewater was treated to recover W via acid precipitation, followed by thermal oxidation using sodium hypochlorite (NaOCl) to dismantle organic complexes, which facilitates subsequent Ni recovery through alkali precipitation [10].

With respect to the treatment of tungsten ores, the pyro- and hydrometallurgical processing of a wolframite concentrate to recover ammonium paratungstate (APT) had been described [11]. In this work, the use of $NaHSO_4 \cdot H_2O$ to roast wolframite, as an alternative to, i.e., sodium carbonate, and water leaching of the roasted product served as the base to produce APT.

A combination of pyro- and hydrometallurgical processing of spent selective catalytic reduction catalysts was conducted to recover tungsten, vanadium, and titanium [12]. In the process, vanadium was first separated into the raffinate of a liquid–liquid extraction operation, followed by titanium and, lastly, tungsten.

W-Mo bimetallic sulfides ($Mo_{1-x}W_xS_2$ nanoflowers) were produced from the treatment of a secondary waste (W-Mo alloy scrap) via a hydrothermal process [13].

Scheelite ores had also been the subject of recent investigations aimed at the recovery of tungsten. A series of scientists considered that heap leaching technology has a series of advantages (i.e., minimum operating and capital costs) over agitation leaching. Thus, in [14], the performance of this heap leaching technology has been increased by the pelletization of the scheelite ore originating from the Republic of Kazakhstan. A mixture of hydrochloric and oxalic acids is used as a leachant for scheelite. Against the above, agitation leaching on a synthetic scheelite sample was investigated using a mixture of sulfuric acid and $H_2O_2$ to dissolve tungsten [15].

Tungsten tailings were the target for the recovery of tungsten; among the various procedures, phytoremediation [16] and gravity separation [17] were used.

As an alternative to the different procedures described above, solvent extraction with TBP (tributyl phosphate) is used to extract tungsten from sodium tungstate solutions. Further, water stripping can efficiently recover tungsten to produce a metatungstic acid solution [18]. Another extractant, in this case, the quaternary ammonium salt Aliquat-336, but in sulfate cycle instead of the most ordinary chloride cycle, was used in the separation of V(V), W(IV), and As(V) from a real leachate [19].

Also, adsorption methodologies were used for the recovery of tungsten [20,21] or to investigate the W(VI)-Mo(VI) separation [22].

The present work investigates the recovery of tungsten from three concentrates of Spanish origin (see below). Two of these concentrates contained scheelite or wolframite as the main species, whereas the third consisted of a scheelite–wolframite mixture. Different operational procedures are considered for these concentrates, allowing the obtaining of

tungstic acid or pure calcium tungstate (synthetic scheelite) as final products, which, on the other hand, can serve as precursors for the production of other tungsten species (ammonium paratungstate, metallic tungsten).

## 2. Materials and Methods

The concentrates came from a tin deposit located in northwest Spain (Orense province, located in the Galicia region). The tin ore, after mining operations, was subjected to an electrostatic separation operation. This operation resulted in obtaining tin concentrates and various tungsten-bearing concentrates [23], whereas the various concentrates used in this work presented the composition shown in Table 1. All chemicals used in the investigation were of AR grade (HCl CAS: 7647-01-0. Merck, Darmstadt, Germany).

**Table 1.** Tungsten content of the three concentrates.

| Concentrate | % Tungsten | Others |
|---|---|---|
| wolframite | 50 | Sn (2%) |
| scheelite | 23 | Quartz, magnetite, hematite, rutile |
| mixed scheelite–wolframite | 23 | Sn (42%), Pb (0.8%), Fe (3%) |

Information supplied by the tin electrorefining company [24].

The concentrates, as supplied from the ore deposit, presented a certain grade of grinding. These materials were further ground (ball mill), and after, samples of each concentrate were milled in a TEMA mill (Siebtechnik, Cincinnati, OH, USA). These last materials were seized on the various meshes.

Leaching and precipitation experiments were carried out in a 500 mL glass reactor provided with water reflux and were heated and stirred via a magnetic plate. After the filtration of the samples, tungsten was analyzed in the liquid phases by Inductively Coupled Plasma–Mass Spectrometry (ICP-MS) (Perkin Elmer ELAN 6000, Waltham, MA, USA). pH measurements were performed using a Crison pH/mV-meter 506 provided with a combined electrode (Ceison S.A., Alella, Barcelona, Spain).

Though the leaching efficiency (% W) can be calculated in different forms [24,25], we used the following relationship:

$$\%W = \frac{[W]_t}{[W]_T} \times 100 \quad (1)$$

where $[W]_t$ is the tungsten concentration in the solution at an elapsed time and $[W]_T$ is the theoretical tungsten concentration in the solution assuming 100% yield in the operation under the experimental conditions tested in each case.

Surface characterization (tungstic acid and synthetic scheelite) was carried out using a scanning electron microscope with a field emission gun (FEG-SEM) (Hitachi S 4800, Hitachi Europe S.A. (Spain), Madrid, Spain), which was also equipped with energy-dispersive X-ray spectroscopy (EDX) (Oxford Instruments NanoAnalysis, Abingdon, UK). An X-ray diffractogram (XRD) was recorded on a Siemens D500 X-ray diffractometer (Siemens AG, Munich, Germany).

## 3. Results and Discussion

### *3.1. Leaching of Wolframite Concentrate*

#### 3.1.1. Influence of the Leaching Agent

Figure 1 shows the results obtained from the leaching experiments using various NaOH solutions.

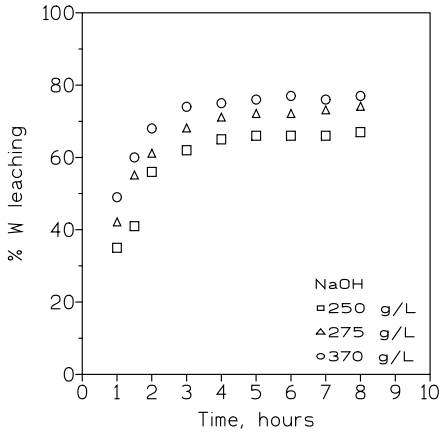



**Figure 1.** Influence of NaOH concentration on tungsten leaching. Temperature: 100 °C. Pulp density: 50 wt% (around 75 g W). Particle size (PS): 105 μm < PS < 150 μm.

The results indicated that the best tungsten dissolution was reached when the most NaOH-concentrated solution (370 g/L) was used in the experiments. As a general rule, a sharp increase in the recovery of tungsten between one and three hours was observed, and after this time, the dissolution proceeded smoothly.

3.1.2. Influence of the Particle Size

The influence of the particle size on tungsten recovery from the wolframite concentrate was also investigated. In this series of experiments, NaOH solutions of 370 g/L were used as a leachant, whereas the other experimental variables were fixed, as shown in Figure 1. The results from this experiment are shown in Figure 2. It can be seen that the decrease in the particle size increased the percentage of tungsten dissolution, though leaching rates of at least 95% were only reached after four hours of reaction time.

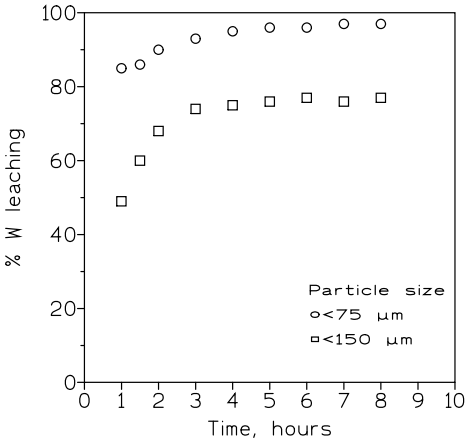



**Figure 2.** Influence of particle size on tungsten leaching. Temperature: 100 °C. Pulp density: 50 wt% (78 g W). NaOH concentration: 370 g/L. Particle size (PS): 53 μm <PS < 75 μm (<75 μm); 75 μm < PS < 150 μm (<150 μm).

The reactions responsible for tungsten dissolution are as follows:

$$FeWO_4 + 2NaOH \rightarrow Na_2WO_4 + Fe(OH)_2 \downarrow \quad (2)$$

$$MnWO_4 + 2NaOH \rightarrow Na_2WO_4 + Mn(OH)_2 \downarrow \quad (3)$$

NaOH in excess is necessary to accomplish the dissolution of this strategic element. From these results, it can be concluded that in the recovery of tungsten from the wolframite

concentrate, the increase in the NaOH concentration in the leaching solution does not have an appreciable effect on the rate of leaching. However, the variation of the m > particle size has a key and positive influence on this leaching rate. Thus, using the lowest particle size, one can yield solutions of about 750 g/L $Na_2WO_4$, serving as a starting point to obtain ammonium paratungstate or synthetic scheelite.

### *3.2. Leaching of the Scheelite Concentrate*

#### 3.2.1. Influence of the Temperature on Tungsten Recovery

These experiments were carried out on samples with a sample of particle size between 53 µm and 75 µm, 200 g/L of HCl, and 3.2 wt% pulp density (3.9 g W). The results of this experiment are shown in Figure 3. It can be seen that an increase in temperature increases the leaching rate; however, in all these cases, the leaching rate is very low.

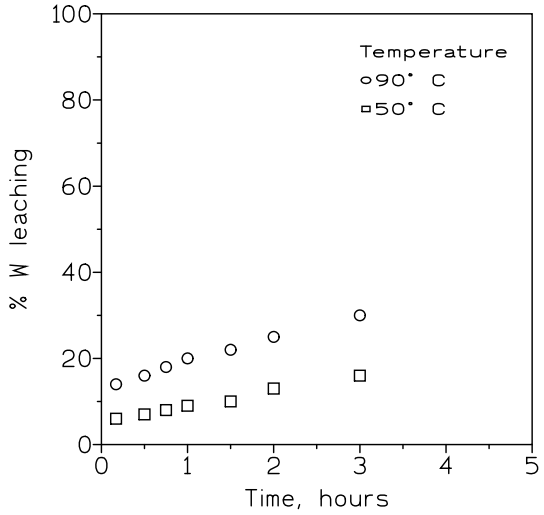



**Figure 3.** Influence of temperature on tungsten recovery from the scheelite concentrate.

#### 3.2.2. Influence of the Particle Size

The variation of the particle size on tungsten recovery was next investigated at 90 °C, using the other same experimental variables as in Figure 3. Figure 4 shows the results of these exoeruments.

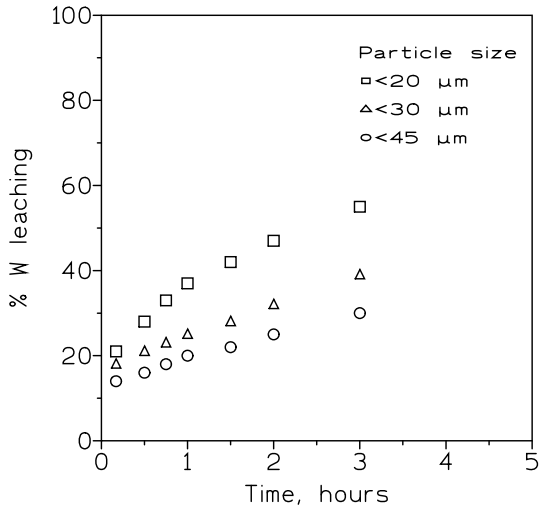



**Figure 4.** Influence of the particle size on tungsten recovery. Temperature: 90 °C. HCl concentration: 200 g/L. Pulp density: 3.2 wt% (3.9 g W). Particle size (PS): PS < 20 µm (<20 µm); 20 µm < PS < 30 µm (<30 µm); 30 µm < PS < 45 µm (<45 µm).

As in the case of the wolframite concentrate, the variation of the particle size of the scheelite concentrate has a key effect on tungsten recovery. This rate increased with the decrease in the particle size; this influence is more noticeable when longer reaction times are used.

#### 3.2.3. Influence of the Leachant on Tungsten Recovery

In order to gain information about the leaching behavior of this concentrate when different HCl concentrations are used to leach tungsten, a series of experiments were performed on a sample with a particle size between the 20 μm and 38 μm range. Figure 5 shows the results obtained at the various HCl concentrations.

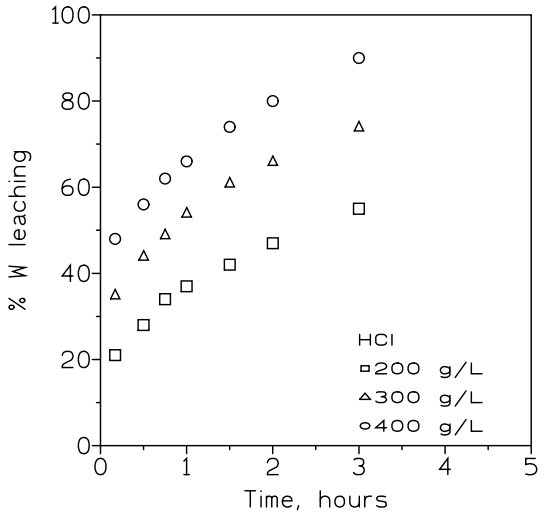



**Figure 5.** Influence of the HCl concentration on tungsten recovery. Temperature: 90 °C. Pulp density: 3.2% wt (around 4.3 g W).

These results indicated that the leaching rate of tungsten increased with the increase in the HCl concentration, with near complete dissolution of the concentrate using 400 g/L HCl as a leachant and three hours.

In the case of this concentrate, tungsten dissolves according to the following reaction:

$$CaWO_4 + 2HCl \rightarrow H_2WO_4 + CaCl_2 \quad (4)$$

As a general conclusion, the change in temperature is the variable that least affects the tungsten dissolution process from this scheelite concentrate. Similar to the wolframite concentrate, it is necessary to use an excess of the leachant, in this case HCl, to reach adequate tungsten recoveries from the scheelite concentrate; though in this case, the utilization of this excess can be attributed to the formation of a tungstic acid film around the scheelite particle, impeding the subsequent attack [26,27].

### *3.3. Leaching of a Scheelite–Wolframite Concentrate. Acidic Medium*

#### 3.3.1. Influence of the HCl Concentration on Tungsten Leaching

The influence of the HCl concentration on tungsten recovery from the mixed concentrate is shown in Table 2.

**Table 2.** Percentages of the leaching of tungsten from the mixed concentrate.

| [HCl], M | Temperature, °C | Two Hours | Four Hours |
|---|---|---|---|
| 6 M | 20 | 0 | 0.05 |
| 6 M | 40 | 0.08 | 0.1 |
| 6 M | 80 | 0.2 | 0.3 |
| 12 M | 20 | 0.04 | 0.8 |
| 12 M | 40 | 1 | 1 |
| 12 M | 80 | 56 | 69 |

Pulp density: 16.7 wt% (28 g W in the case of using 12 M HCl solutions).

These results show the ineffectiveness of hydrochloric acid for the dissolution of the concentrate. Only in the case of using concentrated acid (12 M) and a high temperature (80 °C) is a yield above 50% obtained.

3.3.2. Influence of the Pulp Density on Tungsten Recovery

This variable was also investigated, and the results are summarized in Table 3.

**Table 3.** Percentage of the leaching of tungsten at various pulp densities.

| Time, Hours | 16.7 wt% (28 g W) | 8.4 wt% (14 g W) | 4.2 wt% (7 g W) |
|---|---|---|---|
| 2 | 56 | 21 | 14 |
| 4 | 69 | 23 | 14 |

HCl concentration: 12 M. Temperature: 80 °C.

The variation of the pulp density influenced the percentage of tungsten recovered in the solution; thus, the use of more diluted slurries causes the percentage of tungsten recovery to decrease drastically. Except in the case of the more concentrated slurry, the variation of time (2–4 h) does not influence the recovery rate.

### *3.4. Leaching of a Scheelite–Wolframite Concentrate. Alkaline Medium*

Due to the ineffectiveness of the acid attack on the mixed concentrate, experiments were carried out using NaOH solutions. Different pulp densities were used in this experiment, and the results are summarized in Table 4.

**Table 4.** Influence of the pulp density on tungsten recovery.

| Time, Hours | 16.7 wt% (28 g W) | 8.4 wt% (14 g W) | 4.2 wt% (7 g W) |
|---|---|---|---|
| 2 | 6 | 8 | 10 |
| 4 | 9 | 13 | 14 |
| 6 | 10 | 16 | 18 |
| 8 | 12 | 21 | 21 |

NaOH concentration: 400 g/L. Temperature: 100 °C.

### *3.5. Pyro-Hydrometallurgical Treatment of the Mixed Concentrate*

Due to the ineffectiveness of the acid and alkaline attacks on the concentrate, a mixed pyro-hydrometallurgical procedure was investigated. As a first step, the concentrate was roasted in sodium carbonate media and various temperatures, and the roasted material was further subjected to a leaching procedure using water. The results from these two steps are described below.

3.5.1. Roasting of the Concentration in Sodium Carbonate Media

Table 5 summarizes the various experiments carried out on the mixed concentrate.

**Table 5.** Tests of roasting on the scheelite–wolframite concentrate.

| Test | Temperature, °C | Time, Hours | Excess $Na_2CO_3$, % |
|---|---|---|---|
| 1 | 600 | 2 | 30 |
| 2 | 600 | 1 | 40 |
| 3 | 600 | 2 | 40 |
| 4 | 700 | 2 | 20 |
| 5 | 700 | 2 | 30 |
| 6 | 800 | 2 | 40 |
| 7 | 800 | 1 | 20 |
| 8 | 800 | 2 | 20 |
| 9 | 800 | 1 | 30 |
| 10 | 800 | 2 | 30 |
| 11 | 800 | 1 | 40 |
| 12 | 800 | 2 | 40 |
| 13 | 800 | 2 | 20 |
| 14 | 800 | 2 | 40 |

Test 1–12: particle size < 150 μm. Tests 13 and 14: particle size < 231 μm.

The reactions occurring during this roasting step are as follows:

$$4FeWO_4 + O_2 + 4Na_2CO_3 \rightarrow 4Na_2WO_4 \downarrow + 2Fe_2O_3 \downarrow + 4CO_2 \quad (5)$$

$$6MnWO_4 + O_2 + 6Na_2CO_3 \rightarrow 6Na_2WO_4 \downarrow + 2Mn_3O_4 + 6CO_2 \quad (6)$$

$$CaWO_4 + Na_2CO_3 \rightarrow Na_2WO_4 \downarrow + CaO \downarrow + CO_2 \quad (7)$$

In the case of the iron–manganese species, the presence of an oxidant (air) is necessary to achieve the complete oxidation of both metals. In the case of Equation (6), the formation of CaO is undesirable since in the subsequent leaching stage, the presence of this oxide can promote the formation of solid scheelite with the subsequent loss of tungsten. This reaction can be avoided by adding silica during the roasting step so that an insoluble calcium silicate is formed, preventing the formation of scheelite.

3.5.2. Leaching of the Roasted Materials

As is said above, the leaching experiments of the roasted materials were carried out using water as a leachant, and the influence of the pulp density was investigated. Table 6 summarizes the results from these series of experiments.

**Table 6.** Percentage of tungsten recovery on the roasted materials.

| Test (With Respect to Table 4) | Pulp Density, wt% | % W Leached |
|---|---|---|
| 1 | 25 (29 g W) | 71 |
| | 16.7 (23 g W) | 75 |
| | 12.5 (17 g W) | 80 |
| 5 | 25 (29 g W) | 75 |
| | 16.7 (23 g W) | 80 |
| | 12.5 (17 g W) | 84 |

**Table 6.** *Cont.*

| Test (With Respect to Table 4) | Pulp Density, wt% | % W Leached |
|---|---|---|
| 10 | 25 (29 g W) | 80 |
| | 16.7 (23 g W) | 86 |
| | 12.5 (17 g W) | 91 |
| 12 | 25 (29 g W) | 94 |
| | 16.7 (23 g W) | 98 |
| | 12.5 (17 g W) | 99 |
| 14 | 25 (29 g W) | 60 |
| | 16.7 (23 g W) | 69 |
| | 12.5 (17 g W) | 73 |

Temperature: 90 °C. Time: 2 h.

These results indicated that to yield better tungsten recoveries, it is necessary the use an excess of sodium carbonate (i.e., 40%) over the stoichiometric reaction and a roasting temperature in excess of 700 °C. As a general rule, the percentage of tungsten recovery increased with the use of more diluted pulps.

### *3.6. Treatment of the Leachates*

#### 3.6.1. Acidic Medium

The use of an acidic medium as a leachant was only effective in the case of the scheelite concentrate (Section 3.2). As a result of this operation, tungstic acid was formed. This compound either precipitates on cooling the filtered solutions [28] or precipitates in the leaching process itself, together with the insoluble residue.

In the second case, a purification of the impure tungstic acid is needed; this step includes the acid dissolution in ammoniacal (concentrated ammonium hydroxide) and 60 °C) or NaOH (50 g/L at room temperature). Further, and using, as in our case, tungstate solutions containing 85 g/L W, the re-precipitation of tungstic acid is carried out using 10 M HCl and in excess with respect to the following reaction:

$$WO_4^{2-} + 2HCl \rightarrow H_2WO_4 + 2Cl^- \quad (8)$$

Temperature also has an influence on this re-precipitation; a high temperature (i.e., 80 °C) is beneficial with respect to 25 °C. Moreover, the solid obtained at this last temperature has a white color, while that obtained at 80 °C has the characteristic yellow color of tungstic acid. The white precipitate contained some impurities, which need to be eliminated by successive washing with dilute acidic (HCl) solutions and/or re-precipitation operations.

Of the two options discussed at the beginning of this section, in the first option, let us recall that the precipitation of tungstic acid by cooling the filtered solution after the acid leaching stage allowed us to yield pure yellow tungstic acid. Next, Figure 6 shows a Scanning Electron Microscopy (SEM) image of the yellow tungstic acid derived from the above procedure. Figure 7 showed the EDX analysis on the zone marked with the black square.

Finally, the re-precipitation process benefits from adding the hot tungstate solution to the hot HCl medium, and not the other way around. Also, the addition of the tungstate solution at a high speed over the HCl solution favors the formation of coarser particles of tungstic acid.

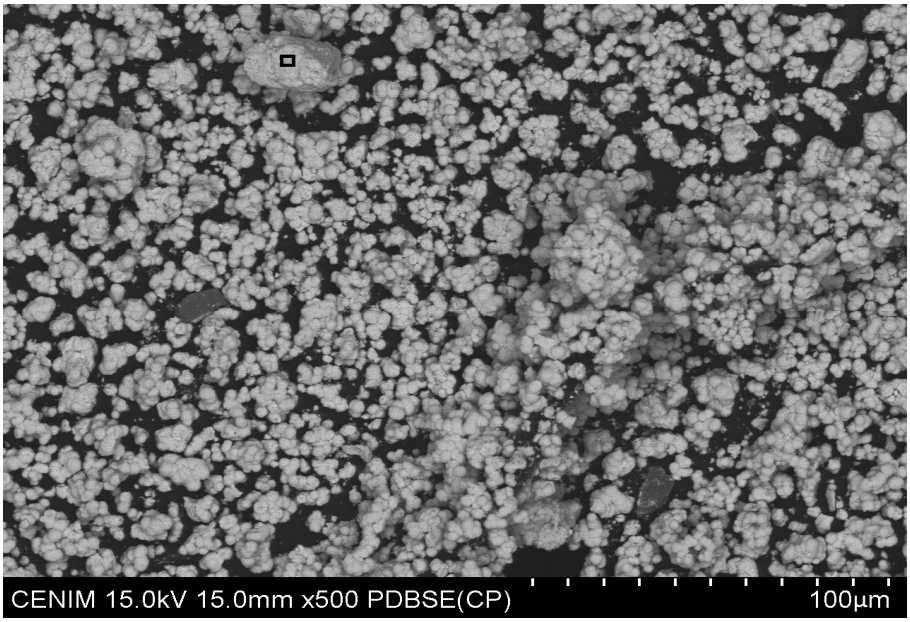



**Figure 6.** Backscattered electron SEM image of tungstic acid.

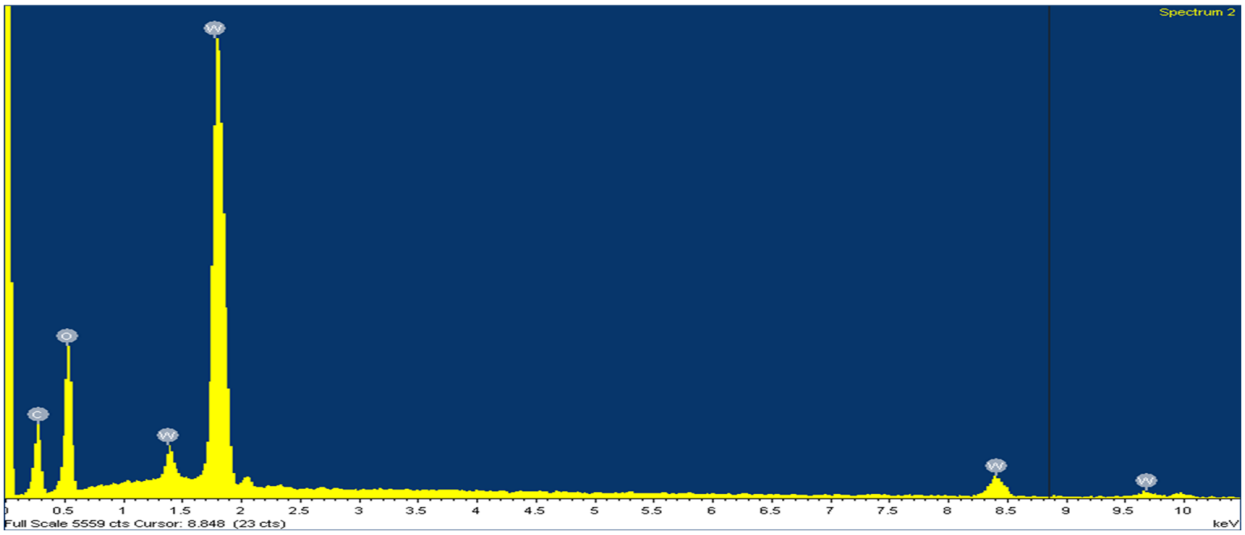



**Figure 7.** EDX of Figure 6: oxygen (27.89% weight, 81.63% atomic) and tungsten (72.11% weight, 18.37% atomic). Analysis carried out on black square of Figure 6.

### 3.6.2. Alkaline Medium

Since the wolframite or the scheelite–wolframite mixture dissolution process yielded alkaline sodium tungstate solutions, the beneficiation process for these solutions included the obtaining of pure calcium tungstate (synthetic scheelite).

In the present investigation and starting from sodium tungstate solutions of about 10 g/L W and pH values greater than 8, the precipitation was experimented with using $CaCl_2$ solutions (though the use of $Ca(OH)_2$ was also acceptable), following the next reaction as follows:

$$Na_2WO_4 + CaCl_2 \rightarrow CaWO_4 \downarrow + 2NaCl \quad (9)$$

The precipitation process was carried out at temperatures in the 15–40 °C range (Figure 8), showing that an increase in temperature favored the removal of tungsten from the solution, and thus, the precipitation of synthetic scheelite.

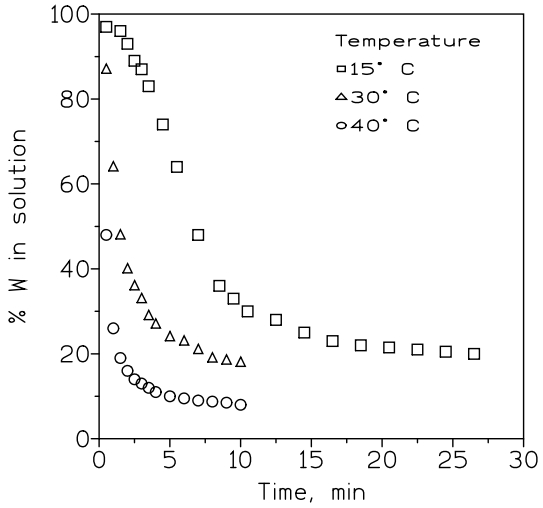



**Figure 8.** Removal of tungsten (precipitation of synthetic scheelite) from sodium tungstate solutions at various temperatures.

With respect to the obtaining of this scheelite, the best yields are obtained by maintaining the tungsten solution at a pH value higher than 8 and using a concentration of $CaCl_2$ 15% higher than that corresponding to the stoichiometry of the previous reaction. A finer particle size solid is obtained by adding the calcium solution on top of the tungsten solution.

Figure 9 shows an SEM image of the synthetic scheelite obtained at 40 °C, whereas Figure 10 shows the corresponding EDX analysis. Figure 11 shows the XRD spectrum of the same sample. In the last figure, it can be seen that the solid has crystallized in the tetragonal system.

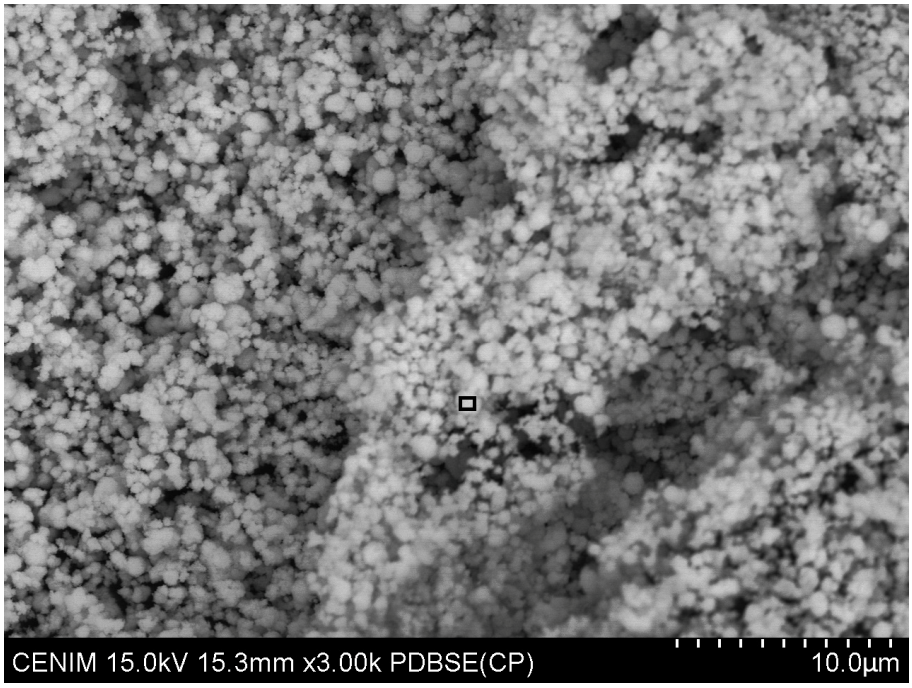



**Figure 9.** Backscattered electron SEM image of the synthetic scheelite.

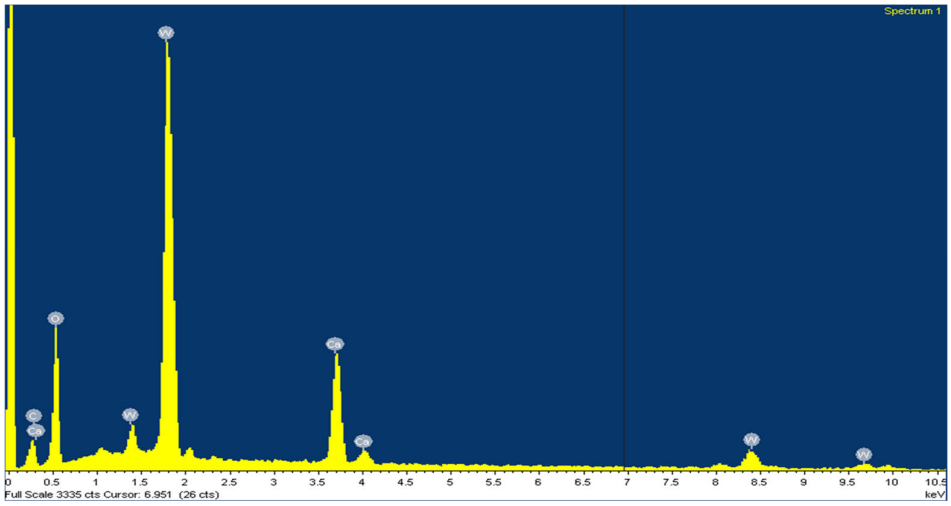



**Figure 10.** EDX of Figure 9: oxygen (28.04% weight, 73.17% atomic), calcium (12.87% weight, 13.41% atomic), and tungsten (59.09% weight, 13.42% atomic). Analysis carried out on the black square of Figure 9.

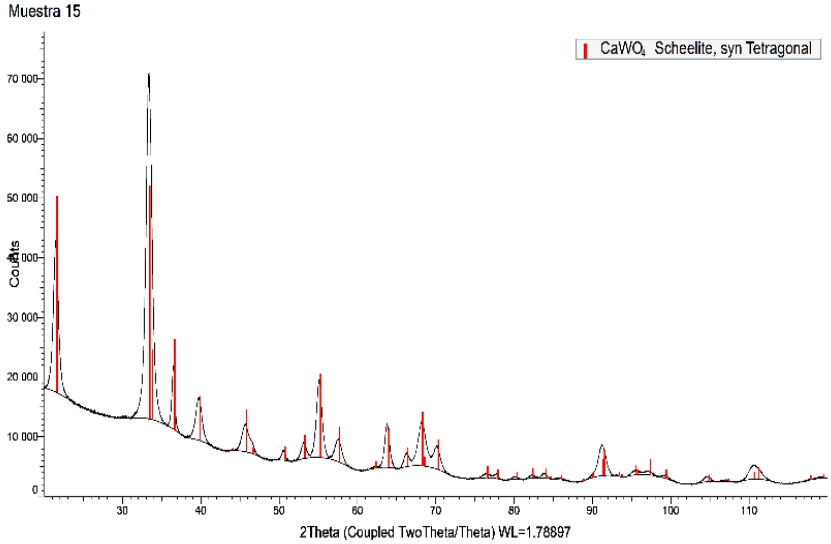



**Figure 11.** XRD spectrum of the synthetic scheelite (Figure 9).

### *3.7. General Flowsheets*

Based on the results presented in the previous sections, the treatment of each concentrate differs from one another; thus, particular flowsheets may be generated for each concentrate. These are presented in Figures 12–14 for wolframite, scheelite, and mixed wolframite–scheelite concentrates.

It is worth mentioning that all these schemes are flexible enough to allow the inclusion of stages to improve the quality of the products (especially intermediates). Likewise, the final products (tungstic acid, synthetic scheelite) can be used to obtain other products of interest, such as APT, $WO_3$, or metallic tungsten, though this investigation has not gone to these extremes.

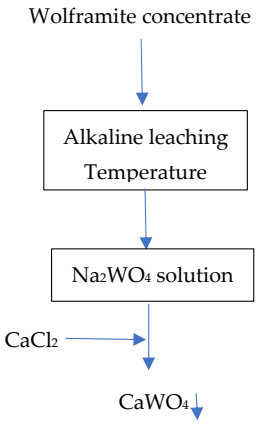



**Figure 12.** General flowsheet for the hydrometallurgical treatment of the wolframite concentrate.

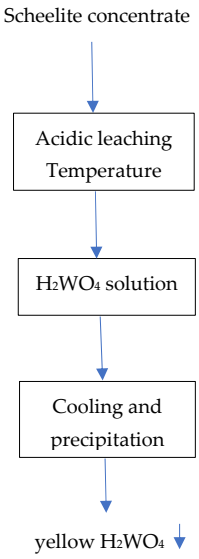



**Figure 13.** General flowsheet for hydrometallurgical treatment of the scheelite concentrate.

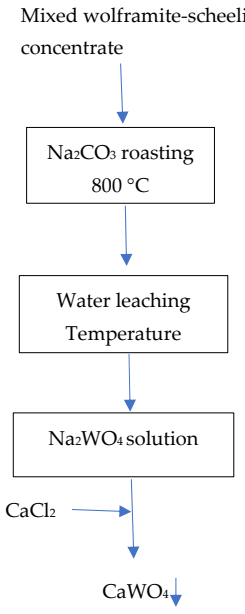



**Figure 14.** General flowsheet for the pyro-hydrometallurgical treatment of the mixed wolframite-scheelite concentrate.

## 4. Conclusions

In this work, we have developed a series of processes to obtain tungsten from concentrates of Spanish origin. The characteristics of these concentrates make these processes differ from each other; the best conditions for obtaining a pure tungsten product (tungstic acid or synthetic scheelite) are indicated below.

Wolframite concentrate: alkaline leaching 370 g/L NaOH, 100 °C, particle size < 75 µm, four hours, and pulp density 50 wt%. Scheelie concentrate: acidic leaching (400 g/L HCl), 90 °C, particle size < 20 µm, 3–4 h, and pulp density 3.2 wt%. Mixed wolframite–scheelite concentrate: (1) alkaline roasting (excess of sodium carbonate, 800 °C, two hours; (2) water leaching of the roasted material (90 °C, two hours, pulp density 12.5–16.7 wt%).

As a general rule, the results obtained show that an increase in temperature and a decrease in particle size favor higher leaching efficiency.

From the leachates, both tungstic acid and synthetic scheelite are obtained experimentally. Both formulations open the way to yield smart tungsten compounds (ammonium paratungstate, tungsten(VI) oxide, or metallic tungsten).

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