

# Techno-Economic Analysis of Electrochemical Recovery of Metals from Waste Printed Circuit Boards (PCBs)

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## Abstract

Electronic waste (e-waste) is a potential source of valuable and critical materials such as gold, silver, platinum group metals, copper, and rare earth elements. In Canada, however, less than 20% of generated e-waste is recycled. There exists a need for an efficient, sustainable e-waste metal recovery process that could be deployed on a regional scale to incentivize collection and recycling. In this paper, a process model was developed in MATLAB to simulate the electrochemical recovery of base and precious metals from printed circuit boards (PCBs) using regenerated  $\text{Fe}^{3+}$  oxidant. In the modelled process,  $\text{Fe}^{3+}$  ions are regenerated at the anode of an electrolytic cell while extracted metals are recovered on the cell cathode. A two-stage process using unique electrolyte solution for each allows base metals (Cu, Sn) and precious metals (Ag, Au, Pd) to be recovered separately. The process model was able to predict economic and environmental performance of the process given a set of feed conditions and operating parameters. Monte Carlo analysis and sensitivity analysis were used to determine key operating parameters and optimal setpoints for a final design. The final design was environmentally and economically feasible, with an estimated net annual profit of CA\$ 321,000 after tax, and a carbon intensity 57.7% lower than the conventional pyrometallurgical process used for e-waste metal recovery.

## Keywords

E-waste, recycling, electrometallurgy, critical metals, process modeling

## 1. Introduction

Electronic waste (e-waste) consists of various types of electronic equipment which have been discarded. E-waste is the fastest growing waste stream in the world; this growth can be attributed to widespread adoption of electronics across the globe and a relatively short lifespan for many of these products (Bel et al. 2019). E-waste streams present significant potential environmental and health harms because they contain many hazardous materials which may seep into the surrounding environment if they are disposed of improperly. These hazardous components include lead, cadmium, mercury, and hazardous organic compounds like flame retardants (Islam et al. 2019). At the same time, e-waste presents an economic opportunity; a 2019 report released by the Platform for Accelerating the Circular Economy estimated the material value of global e-waste accumulation at € 55 billion (Bel et al. 2019). The rise of electronics and e-waste has coincided with a decline in the availability of mineral resources needed to produce electronics (Kumar and Holuszko 2016). These same valuable and scarce resources are present in much higher concentrations in e-waste than they are in ores extracted from mines, meaning recycling of e-waste has the potential to be much more efficient. An estimated 60-95% of energy usage could be reduced by recovering metals from e-waste versus extracting the metals from the earth using traditional methods (Khaliq et al. 2014).

Canada is a large producer of e-waste, producing 725 kilotonnes in 2014 (Kumar and Holuszko 2016). Through a variety of provincial recycling programs and initiatives, an estimated 14% of this e-waste is collected annually (Kumar and Holuszko 2016). The low collection rate for e-waste in Canada is in part due to an insufficient capacity for e-waste processing and a lack of economic incentive for collection – especially the collection of smaller e-waste streams. In Canada, the conventional process for metal recovery from e-waste employs pyrometallurgical smelting units. One pyrometallurgical recycling plant, the Glencore Copper Horne Smelter in Quebec, processes over 100,000 tonnes of e-waste material annually (Glencore Recycling 2020). While these plants individually have a large capacity for e-waste processing, they also have high energy intensities and emissions rates. Additionally, they are generally not economical below an estimated 30,000 tonnes of waste per year due to a high operating and capital costs (Ghodrat et al. 2016). Relying on a centralized collection scheme directing e-waste to large-scale pyrometallurgical plants will inevitably allow a large fraction of smaller e-waste streams to “slip through the cracks”.

There is a clear need for a more localized, lower emission electronic waste processing solution to supply metals economically and provide a greater incentive for the collection of e-waste streams. This paper considers a process design for the electrochemical recovery of base and precious metals from discarded printed circuit boards (PCBs). PCBs are found in virtually all electronics and contain significant amounts of valuable and critical minerals (Bizzo et al. 2014). Canada has recently designated many of these minerals as critical metals of strategic importance; this designation encompasses platinum group metals, copper, tin, aluminium, and rare earth elements (REEs) (Government of Canada 2021).

### 1.1 Objectives

The two primary goals for this paper were to develop a process model for the electrochemical extraction and recovery of base and precious metals from waste PCBs and to employ this model to determine an optimal set of design parameters. The model was designed to have tunable operating parameters allowing for the determination of optimal setpoints based on economic and environmental criteria and constraints. Feasible design cases were identified where CO<sub>2</sub> emissions were lower than the conventional pyrometallurgical route in Canada (< 0.411 t CO<sub>2</sub>e/t mass recovered), and the net annual profit allowed a payback period under 10 years. The success criteria used for optimization were waste recovery efficiency per mass recovered, emissions per mass recovered, net annual profit after tax, and value recovery efficiency.

## 2. Literature Review

Printed circuit boards contain a wide variety of metals which are generally split into three categories: base metals, precious metals, and rare-earth elements (REEs) (Bizzo et al. 2014). Base metals are typically abundant and inexpensive metals such as copper, iron, tin, lead and aluminium. Despite their relative abundance, many of these materials are nonetheless identified as critical minerals by the Government of Canada due to their strategic importance in the economy (Government of Canada 2021). Precious metals such as gold, silver, and palladium are not nearly as abundant and thus are much more economically valuable. REEs include the lanthanide series of elements plus scandium and yttrium. Many of these elements have specific applications in electronics and are also identified as critical minerals (Government of Canada 2021). REEs are generally found only in extremely low concentrations in ore deposits relative to other metals (Lister et al. 2014). PCBs are a potentially critical source of metals not only because they contain a wide array of minerals but also because they contain a significantly higher concentration of metals than is usually found in ore deposits. For example, PCBs contain 12-29 wt.% Cu compared to 0.5-3 wt.% found in ore and 0.01-0.52 wt.% Ag compared to <0.001 wt.% found in ore (Bizzo et al. 2014). Table 1 shows the average metallic composition of PCBs based on a collection of studies done between 2008 and 2014 (Bizzo et al. 2014).

Table 1. Average metallic composition of PCBs from 6 studies conducted between 2008-2014 (Bizzo et al. 2014).

Metal	Composition (wt%)	Metal	Composition (ppm)
Cu	19.36	Ag	167.00
Al	2.01	Au	74.33
Pb	1.49	Cd	197.17
Zn	0.60	K	30.00
Ni	1.39	Mn	680.17
Fe	2.94	Se	3.50
Sn	2.31	As	1.83
Cr	0.06	Pd	36.67

E-waste recycling typically occurs in 2 main steps – primary recycling and secondary recycling. The former physically disassemble the collected waste and sort it into various streams including pure steel, plastics, ink, and PCBs. After this, e-waste streams can be sent to specific secondary recycling processes for further extraction and recovery of materials (Kumar and Holuszko 2016). The scope of this paper was a design for a secondary recycling process for the recovery of base and precious metals from PCBs.

Currently, the most widely used secondary recycling methods for metal extraction from e-waste are pyrometallurgical processes (Khaliq et al. 2014). These processes employ extremely high temperatures to separate and extract both base metals and precious metals from waste streams. Examples of pyrometallurgical operations include smelting in furnaces, incineration, melting, sintering, and gaseous-phase reactions (Ghodrat et al. 2016). These separation processes typically create solvent metal phases; each component metal concentrates in a phase based on its metallurgical properties (Khaliq et al. 2014). Pyrometallurgical routes are favored in industry for their higher recovery and economic efficiencies. The high plastics content of many e-waste streams such as PCBs is a potential source of energy for pyrometallurgical processes, organic components of e-waste substitute for coke as fuel in smelting. Pyrometallurgical processes typically suffer from some drawbacks, however. Among these is the large capital investment required to develop a smeltery process plant. In an economic analysis of the black copper smelting route for processing e-waste, Ghodrat et al. found that the minimum processing requirement was 30 000 tonnes of waste per year for a plant to be economically viable (Ghodrat et al. 2016). In addition to the high capital costs, pyrometallurgical processes inherently generate a large amount of greenhouse gas emissions. For example, the carbon footprint of the Horne smelter was estimated at 0.411 tonnes-CO<sub>2</sub>/tonne-product (Xstrata Copper Canada 2009). The release of emissions also creates a requirement for off-gas treatment since halogenated flame retardants are present in the e-waste, creating toxic dioxins and furans upon combustion (Jadhav and Hocheng 2015). In general, the extreme temperatures and large capital cost make the pyrometallurgical routes less favourable for a smaller-scale sustainable e-waste recycling plant.

The other major portion of industrial metal extraction methods are hydrometallurgical. Typically, the first step in a hydrometallurgical process is to dissolve powdered/crushed ores or e-waste into a chemical leaching agent. For the extraction of base metals (primarily copper), common leaching solutions are H<sub>2</sub>SO<sub>4</sub>, HNO<sub>3</sub>, and aqua regia using H<sub>2</sub>O<sub>2</sub>, Cl<sub>2</sub>, O<sub>2</sub>, or Fe<sup>3+</sup> as oxidants (Cui & Anderson, 2016). Oxidants are necessary for oxidizing metallic components into soluble cations. The strength of the oxidant and leaching solution determines the extent to which metals are extracted; Fe<sup>3+</sup>, for example, will typically only oxidize base metals, whereas strong oxidizing agents such as Cl<sub>2</sub> may oxidize the noble precious metals as well. After extraction of base metals from PCBs, copper will typically be purified in an electrowinning process where it is deposited on the cathode of an electrochemical cell.

Precious metals such as Ag, Au, and Pd are more noble than base metals and thus less easily oxidized. Generally, the leaching solutions used for precious metals are not selective and will concentrate all three. Aqua regia, a mixture of HCl and HNO<sub>3</sub>, is often employed when high recovery efficiencies for Au are desired; however, due to its instability it loses its effectiveness relatively quickly and is also highly corrosive, a safety and environmental concern (Sethurajan et al. 2019). Cyanide is the most widely used reagent for gold extraction from ores and from primary sources, however it poses some serious environmental concerns due to its toxicity. As an alternative, recent studies have considered thiourea and thiosulfate as potential leaching agents for precious metals. These compounds are advantageous because they are non-toxic while still being relatively cheap, however large quantities of the chemical are generally required to recover a reasonable percentage of gold (Jing-ying et al. 2012).

A concern with hydrometallurgical processes and reason that they are not more widespread in industry is their large demand for chemicals which are often toxic (Khaliq et al. 2014). A subcategory of these processes attempts to solve this issue by applying enhanced electrochemical methods for extraction and recovery. In these processes, oxidants are regenerated in an electrochemical cell, conserving the need for feed chemicals, which creates a more sustainable process (Ghosh et al. 2015). Electrochemical recovery methods also allow for the use of electricity as an energy input which could be sourced from renewables, as opposed to relying on fuel combustion for energy as in pyrometallurgical processes. Diaz et al. propose an electrochemical recovery method where Fe<sup>3+</sup> ions are used as an oxidant in an HCl FeCl<sub>3</sub> electrolyte. Fe<sup>3+</sup> ions are generated at the anode of an electrolytic cell and pumped through a packed bed containing shredded PCBs. The ferric ions oxidize base metals from the PCB which are then recovered through plating on the cathode of the cell (Diaz et al. 2017). Figure 1 below shows a schematic of an electrochemical extraction and recovery process for metals in PCBs.

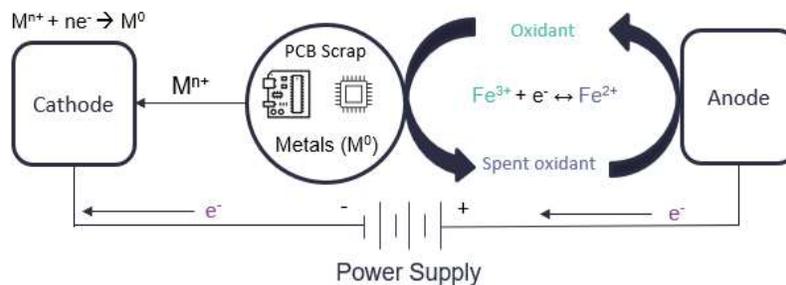
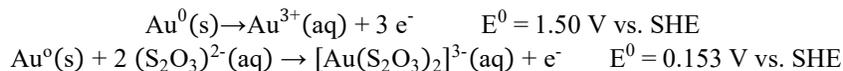


Figure 1. Electrochemical extraction and recovery process for PCB scrap

Other studies have demonstrated the extraction and recovery of precious metals from PCBs using a similar  $\text{Fe}^{3+}$  electrochemical process (Oraby and Jeffrey 2010). While the extraction reactions for precious metals have higher reduction potentials and therefore cannot be oxidized by  $\text{Fe}^{3+}$ , the use of a thiosulfate electrolyte allows for the extraction of precious metals by  $\text{Fe}^{3+}$  into metal-thiosulfate complexes since the reduction potential for the extraction reaction is lowered. For example, the oxidation of gold metal into  $\text{Au}^{3+}$  has a standard Nernst potential of 1.50 V vs. the standard hydrogen electrode (SHE); when reacting with thiosulfate to form a gold-thiosulfate complex, this potential is lowered to 0.153 V vs. SHE (Walting 2007):



### 3. Methods

MATLAB programming software was used to develop a process simulation to calculate economic and environmental performance metrics given a set of feed characteristics and operating parameters. The feed stream was taken to be 200 tonnes/year of PCBs which have been manually separated from other e-waste and roughly shredded by a primary recycler. The PCBs were assumed to have a metal composition as per Table 1 while the remaining fraction was assumed to be an inert non-metal component.

#### 3.1 Pre-processing Model

The first portion of the model simulated the grinding of PCBs into particles of under 1 cm; the exact grinded particle diameter was a tunable operating parameter. It was assumed that the grinded particles were of uniform diameter. Following grinding, non-metal particles were separated through an electrostatic separator model with a 98% separation efficiency as per a similar process unit in literature (Wu et al. 2008). Following electrostatic separation, magnetic separation was modelled to separate the ferro-magnetic portion of the feed, which contained iron, nickel, and a small portion of rare earth elements (REEs). Efficiency of the magnetic separation drum was taken to be 95% as per a similar separation in literature (Chehade et al. 2012). The remaining non-magnetic particle composition and flowrate were then fed into the extraction and recovery process model.

#### 3.2 Extraction and Recovery Process Model

The metal extraction and recovery models simulated the two-stage process of leaching and electrowinning of base and precious metals from grinded PCB particles. The first stage simulated the extraction and recovery of base metals from PCBs using  $\text{Fe}^{3+}$  as a regenerated oxidizing agent in  $\text{HCl-FeCl}_3$  electrolyte. The second stage simulated the subsequent extraction and recovery of precious metals using  $\text{Fe}^{3+}$  oxidant in a thiosulfate- $\text{Fe}(\text{EDTA})$  solution. In both stages, crushed PCBs from the pre-processing outlet are loaded into a leaching vessel, and the electrolyte containing  $\text{Fe}^{3+}$  is circulated through the vessel and an electrowinning cell in series with the vessel. After the base metal operation was run for its cycle time (a tunable parameter), residual PCB solids (with final composition and diameter calculated in the model) were transferred to the leaching unit of the precious metal stage. By extracting the base metals initially, the efficiency of precious metal extraction increased because more precious metal surface area was exposed. The process flow diagram of the extraction and recovery models for both stages is shown in Figure 2.

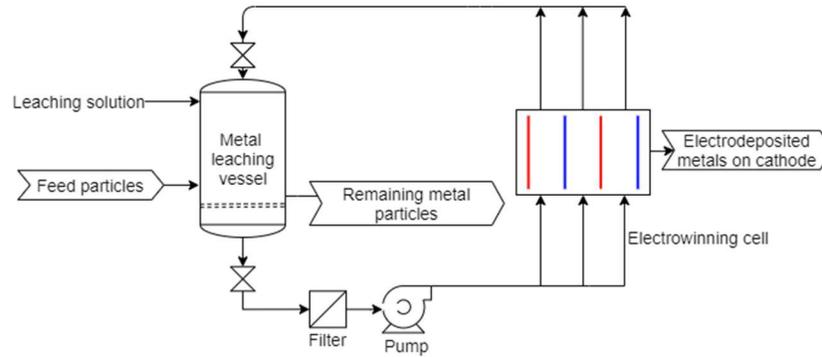


Figure 2. Process flow diagram for the extraction and recovery of both base and precious metals.

PCB mass, particle radius, and composition are process inputs which are fed into each extraction and recovery model stage. For the base metal stage, the PCB input conditions are those computed at the output of the preprocessing step. For the precious metal stage, the PCB input conditions are those computed as the output of the base metal stage. The following operating parameters are set in both the base and precious metal stage of the extraction and recovery model: process unit temperature and pressure; circulation flowrate; cell and vessel dimensions; number and size of electrodes; the applied voltage to the cell; and the processing time. The models output the transient system data at each computed time step in a data structure. These data include ionic concentrations, metal recovered on the cell cathode, metal extracted from PCBs, Nernst potentials for each reaction, and currents for each reaction.

The model simplifies the extraction and recovery process into a system of three well-mixed controlled volumes. One control volume exists in the leaching unit, and two are in the electrowinning cell – the anolyte side and catholyte side. Material balances for each ionic species in each control volume are written to yield a system of ordinary differential equations (ODEs). These ODEs are solved by built-in MATLAB solver *ode15s*. The consumption/generation term for each ionic species consists of the partial currents for each reaction consuming/generating the species. These currents are calculated by solving a system of electrochemical equations that determine electrode potentials and overpotentials for each redox reaction at each time step. Within this system of equations is the Butler-Volmer equation, written below for an oxidation reaction of species A to form species B:

$$i_m = i_{o,m} \left\{ \frac{C_{A,surf}}{C_{A,bulk}} e^{\frac{\alpha_m z_m F (E - E_{rev})}{RT}} - \frac{C_{B,surf}}{C_{B,bulk}} e^{\frac{(1-\alpha_m) z_m F (E - E_{rev})}{RT}} \right\}$$

Where  $i_m$  is the current density for reaction,  $i_{o,m}$  is the exchange current density,  $\alpha_m$  is the charge transfer coefficient,  $z_m$  is the number of electrons transferred,  $T$  is the temperature,  $E$  is the electrode potential,  $E_{rev}$  is the reaction Nernst potential,  $C_{bulk}$  is the concentration of the species in the bulk solution and  $C_{surf}$  is the concentration of species at the electrode surface.  $F$  is Faraday's constant and  $R$  is the gas constant. Current densities can then be multiplied by the available surface area for reaction to determine a current for each reaction. In the leaching bed, the reactive surface area is the surface of reacting metal exposed, while in the electrowinning cell, it is the surface area of the electrodes. In the electrowinning cell, a potential balance can be written:

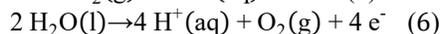
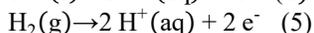
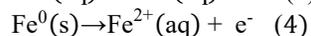
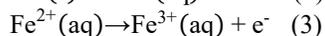
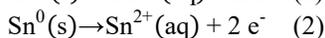
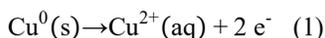
$$V_{app} = E_{cat} - E_{an} + IR$$

Where  $V_{app}$  is the applied voltage to the cell,  $E_{cat}$  is the cathode potential,  $E_{an}$  is the anode potential,  $I$  is the overall current flow, and  $R$  is the electrolyte resistance. The leaching unit is modelled as a short-circuited galvanic cell operating at one electrode potential,  $E_{corr}$ . A final equation in both the leaching vessel and electrowinning cell specifies that the anodic and cathodic current magnitudes must be equivalent for electron transfer. The resulting system of equations are then solved simultaneously for each time step in the ODE solver domain using MATLAB solver *fsolve* for both the leaching vessel and cell.

There were several assumptions made to simplify the model. It was assumed that the process takes place isothermally, and that the electrolyte volume remains constant. It was assumed that sufficient mixture takes place such that the bulk concentrations are uniform. At low reaction overpotentials ( $< 0.15$  V), the surface concentration of ionic species was assumed to be the same as the bulk concentration, while at higher overpotentials ( $> 0.15$  V), mass-transfer limitations on reaction kinetics were considered in computation of the B-V equation. The PCB particles were assumed to be spherical with a radius equal to the average particle radius with all metals distributed uniformly and randomly. It was assumed that Kohlrausch's law of independent ionic migration holds such that conductivity could be computed at infinite dilution. To simplify the model and reduce computation time, it was assumed that current exchange densities and ionic diffusivities would remain constant.

### 3.3 Base Metal Stage Species and Reactions

The base metal extraction and recovery system uses the regenerated oxidant  $\text{Fe}^{3+}$  in a  $\text{HCl-FeCl}_3$  electrolyte solution, based off the experiment done by Diaz et al. (2017). Base metals are extracted while precious metals remain in the PCB particles for later extraction. The ions modelled are:  $\text{Cu}^{2+}$ ,  $\text{Sn}^{2+}$ ,  $\text{Fe}^{2+}$ ,  $\text{Fe}^{3+}$ ,  $\text{H}^+$ , and  $\text{Cl}^-$ . The two base metals considered in the model are thus Cu and Sn. It should be noted that these are not the only base metals in the crushed PCB feed; others, such as iron, nickel, aluminum, and lead, have weight percentages above 0.5% in the feed but contribute negligibly to the overall value of recovered metals. The following half-cell reactions were considered in the base metal model (all written in the anodic direction):

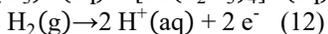
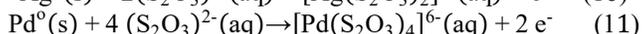
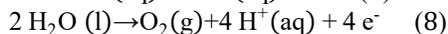
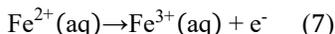


In the electrowinning cell, reaction (3) occurs at the anode as the oxidant,  $\text{Fe}^{3+}$  is regenerated. The desired cathodic reactions are the reverse of (1) and (2). Some  $\text{Fe}^{2+}$  may also be reduced at the cathode as in the reverse of reaction (4). Any precious metal ions in solution may be reduced at the cathode, however these are not expected to occur since the presence of precious metals in the leachate is not expected. Hydrogen evolution (reverse of (5)) may compete for current at the cathode; minimizing the rate of hydrogen generation is important for cell design. Finally, oxygen evolution (reaction (6)) may occur at the anode, however cell parameters should be controlled such that the current efficiency for reaction (5) at the anode is maximized. In the leaching vessel, reaction (3) occurs in reverse as  $\text{Fe}^{3+}$  is reduced to oxidize base metals. Copper and tin leach out of the PCBs as in reactions (1) and (2).

### 3.4 Precious Metal Stage Species and Reactions

This system is designed to recover precious metals such as gold, silver, and palladium, all of which are selectively recovered from a thiosulfate solution with ferric ions and thiourea additives based off a study by Oraby and Jeffrey (2010). The main ions that are modelled and would be present in this solution include:  $\text{Au}(\text{S}_2\text{O}_3)_2^{3-}$ ,  $\text{Ag}(\text{S}_2\text{O}_3)_2^{3-}$ ,  $\text{Pd}(\text{S}_2\text{O}_3)_4^{6-}$ ,  $(\text{S}_2\text{O}_3)^{2-}$ ,  $\text{Fe}^{3+}$ ,  $\text{Fe}^{2+}$ ,  $\text{OH}^-$ ,  $\text{H}^+$ .

The following half-cell reactions were considered in the precious metal model:



Reactions (7) and (8) (oxidant regeneration and hydrolysis) occur in the forward (anodic) direction at the anode of the electrowinning cell. Reactions (9) through (11) occur in reverse (cathodic) direction at the cathode of the electrowinning cell as solid precious metals are plated. A competing hydrogen evolution reaction (12) takes place in the reverse direction at the cathode. In the leaching vessel, reaction (7) takes place in reverse (cathodic) direction

and reactions (9) through (11) occur in the forward, anodic direction as  $\text{Fe}^{3+}$  ions are reduced to  $\text{Fe}^{2+}$  ions to drive extraction, and solid precious metals are extracted to form thiosulfate complexes.

### 3.5 Economic Impacts

For the economic models, equipment costs were estimated from resources from Ulrich and Vasudevan which give the purchased equipment cost as a function of a design parameter (Ulrich and Vasudevan 2004). Electricity costs were calculated by calculating the power requirements of each piece of equipment and multiplying that by the number of annual working hours in a year and the provincial standard electricity rate. The main categories of costs for this project were capital costs, energy costs, chemical costs, and labour/maintenance costs. Capital costs were from the purchase of equipment and vessels, and it was assumed that there would be a 10-year straight line depreciation on these purchases. The energy costs were calculated by the power requirements needed to run the equipment for the required amount of time at a standard provincial energy rate of 10.39¢/kWh (Ontario Energy Board 2021). Chemical costs were calculated based on wholesale bulk chemical costs and the amount of chemicals needed over an annual basis. Additional costs for disposal of the process solution were calculated based on local wastewater disposal costs in Ontario.

The amount of revenue brought in due to selling the metal was calculated by an assumed selling price of 75% of the 2021 market value of metals. This discount factor accounts for the impurity of metals recovered since the base metals and precious metals are each recovered as alloys.

### 3.6 Environmental impacts

The following metrics were identified as the quantitative metrics for evaluating environmental impacts of the system: energy intensity per unit of input,  $\text{CO}_2$  equivalent intensity, water intensity per unit input, and waste recovery.  $\text{CO}_2$  intensity was converted from electricity used based on the 30 g  $\text{CO}_2$  eq/KWh figure reported in Canada's 2020 National Inventory Report (Government of Canada 2020).

## 4. Data Collection

The scripts used to run the simulations (and output technical, economic, and environmental metrics) based on the methods described above were coded using MATLAB, taking advantage of built-in ODE solvers for the transient simulation. Monte Carlo analysis was initially used to reduce the amount of time required for the simulations, as it would enable an analysis of several model parameters at once. In any given simulation, each parameter studied is randomly set based on a given reasonable range and simulated to generate performance results based on that set. This was repeated over many (over 2 000) simulations, with plots of model parameters and performance results being used to identify trends and optimal operating ranges for each studied parameter.

For the model parameters with significant trends in the Monte Carlo analysis, a further sensitivity analysis was conducted to refine and identify optimal parameter points to select for the final design. The final design was selected based on these analyses, using the optimal parameter points, parameter trends and practical engineering judgement to select the parameters used in the model, which was then simulated to identify the resulting performance characteristics.

## 5. Results and Discussion

Based on the analyses conducted, it was found that recovering more base metals reduced the carbon intensity but limited the profitability, while recovering more precious metals increased the profitability but also increased the carbon intensity. This is due to the relative values of precious and base metals being different, with base metals not being as economically profitable to recover compared to precious metals, and although precious metals are more profitable for recovery, they are also more energy intensive to recover a given amount of them.

Two of the most notable parameters for the system were the initial ground PCB particle diameter as well as the residence time in the leaching and electrowinning vessels. The finer the initial PCB particle diameter, the more surface area available for the leaching process to operate, which was a limiting factor for base metal recovery. In effect, this would improve the recovery of the base metals as well as the precious metal recovery downstream due to the increased proportion and area available for the precious metal leaching unit. The residence time was also very important as it affected the rate of leaching and electrowinning and needed to be kept fairly small so as to limit the

amount of iron deposition on the anode while ensuring sufficient leaching and desired electrodeposition on the cathode.

### 5.1 Final Design and Simulation

Based on the Monte Carlo analysis, the sensitivity analysis and subsequent trends discussed previously, selected values were taken for the operating parameters and solution conditions, shown in Table 1 and 2 respectively, in order to have a system that meets the desired characteristics. These selected values were used as the simulation parameters for determining the system performance and characteristics.

Table 2. Selected operating parameters for the final process design

Parameter	Base Metal Value	Precious Metal Value
Grinder output average particle diameter	1 mm	Dependent on base metal output
Applied Voltage	2.5 V	3 V
Cycle Time	72 hrs.	96 hrs.
Recirculation Flowrate	9.57 L/s	6 L/s
Residence Time in Leaching Bed	5 s	2.6 s
Leaching Bed L/D ratio	3	3
Total Leaching Vessel Volume	47.8 L	27.03 L
Electrode Height	0.5 m	1 m
Electrode Length	2 m	1.5 m
Electrode Spacing	4.5 cm	4.5 cm
Number of Electrode Pairs	18	2
Total Cell Volume	1323 L	153 L

Table 3. Initial solution conditions for final selected design

Parameter	Selected value	
<b>Base Metal Stage</b>		
Operating Temperature	25 °C	25 °C
Operating Pressure	1 atm	1 atm
Initial pH	0.3	10
Initial Oxidant (Fe <sup>3+</sup> ) Concentration	1.4 M	0.3 M
<b>Precious Metal Stage</b>		
Initial EDTA Concentration	0.33 M	
Initial Thiourea Concentration	10 mM	
Initial Thiosulfate Concentration	0.1 M	

### 5.1.1 Simulation results

Figure 3 shows how copper concentrations and recovery percentage vary as base metal stage operation proceeds. As shown in the leftmost plot, copper concentration reaches a maximum of approximately 0.58 mol/L after 23 hours of extraction, and then decreases steadily until the end of operation. Effectively all copper is recovered after 60 hours of operation. Similar profiles for tin showed that tin reached a total recovery of approximately 90% through the entire base metal stage cycle.

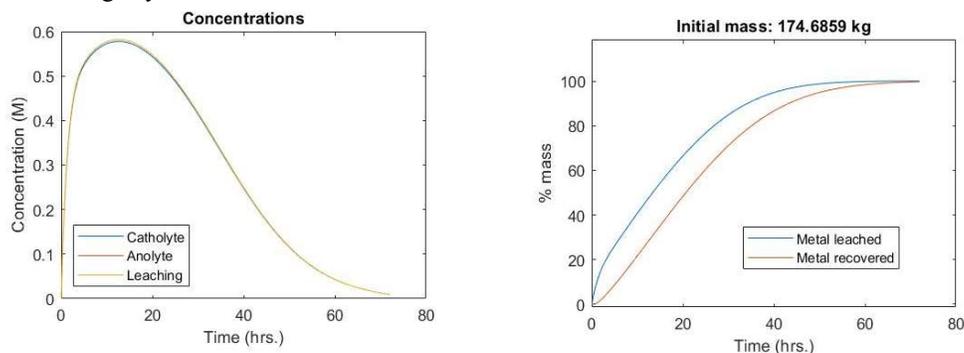


Figure 3. Copper concentration and extraction/recovery profiles during base metal stage operation

A similar trend can be found for the precious metal stage operation, with gold being representative of the precious metals. Gold reaches a maximum concentration of just over 0.0025 mol/L in the electrolyte after approximately 20 hours of leaching, after which the concentration of gold declines; at this point, the rate of recovery exceeds the leaching rate. Gold reaches a total recovery of about 90% after the precious metal stage is complete. Similar plots for palladium and silver show that effectively all the palladium is recovered, while around 2% of the silver remains unrecovered. It is important to note that although not all gold and silver is recovered, effectively all these metals are extracted and dissolved in the electrolyte. As the process is run continuously, using the same electrolyte fluid for each cycle will allow for this extracted material to be recovered in subsequent cycles.

### 5.1.2 Environmental summary

Table 4: Final Design Environmental Results Summary

Metric	Basis	Result	Target
Energy intensity	Throughput basis	1133 kW/t PCBs processed	-
	Mass recovered basis	5813 kW/t metal recovered	-
CO <sub>2</sub> intensity	Throughput basis	0.034 t CO <sub>2</sub> e/t PCBs processed	-
	Mass recovered basis	0.174 t CO <sub>2</sub> e/t metal recovered	0.205 t CO <sub>2</sub> e/t metal recovered
Water intensity	Throughput basis	118.3 L/t PCBs processed	-
	Mass recovered basis	606.9 L/t metal recovered	-
Waste recovery efficiency	Throughput basis	93.3%	50%

While these results indicate that in comparison to the reference process plant (that uses pyrometallurgy) the process designed here is environmentally favorable, there are a variety of ways to modify the design to further reduce the environmental impacts. This includes the reuse of solution over multiple loading cycles, as well as the recovery of metals and reagents from the process wastewater through onsite treatment, if economically viable.

### 5.1.3 Economic summary

Table 5 and Table 6 outline the key economic metrics as generated by the final design. It is clear that the design is feasible, with a payback period of only 1.74 years. The overall capital cost for this process was calculated to be \$642,350 CAD.

Table 5: Final Design Cost Summary

<b>Total capital investment</b>	CAS 642,350
<b>Total annual operating expenses</b>	CAS 970,100

Table 6: Final Design Economic Profitability Summary

<b>Metric</b>	<b>Result</b>	<b>Target</b>	<b>Constraint</b>
Percentage of value recovered from feed	89.1%	> 50%	-
Operating cost percentage of revenue	67.9 %	-	-
Net annual profit after taxes	CAS 321,210	> CAS 115,386	> \$0
Payback period	1.74 yrs	< 5 yrs.	< 10 yrs.

The total revenue generated from the selling of these metals was calculated to be \$1,429,000 CAD per year, resulting in the net annual after tax profit of this process of \$321,210 CAD per year. While the high operating costs and high capital costs could potentially make it hard to secure investments in this project, the high anticipated revenue from this project with a projected payback period of just 1.74 years should make the project enticing enough to attract investors.

While both the main input and output streams for the plant have been roughly specified, validation about these streams' characteristics for costing and potential size from potential suppliers and buyers is likely necessary to be able to have reasonable costing estimates, as well as ensure the long-term viability of such a project. In addition, this understanding is necessary for being able to account for variations in the value and composition of metals, allowing for detailed analysis to identify the impact of such variations on the final system and identify potential recovered metal quality limits to ensure an effective and reliable buyer-supplier relationship. Another related aspect to this is determining the value of the secondary streams from the pre-processing units and identifying alternative uses for them, such as turning them into additional revenue streams to further improve the profitability of the project. For a more detailed design, there may be additional system components that were not fully quantified and could benefit from more detailed costing. Based on the feasibility of the design, there is sufficient leeway in the net profit values to be able to support additional or more expensive equipment that may not have been fully specified in the current economic analysis.

### 5.2 Model Limitations and Scope

As can be seen above, the technical model is successful in modelling the overall system design, allowing the Monte Carlo and the sensitivity analyses to be conducted on key parameters, ultimately determining parameters for the design to meet the constraints and success criteria. While these results are promising, there are some additional considerations.

In general, the model features several simplifying assumptions designed to reduce the overall complexity, such as the pseudo-separate catholyte and anolyte mixtures, as well as neglecting the formation of additional complexes, decomposition of thiosulfate and additional interactive behaviour of the metal ions in solution. These changes were reasonable simplifying assumptions to model the system, but future development will need to further validate the model through focused experimental studies and incorporate these complexities to be able to design and account for any impacts they may have. In particular, the effects of copper and other base metals remaining on the PCB particles after transfer to the precious metal processing stage has not been quantified to reduce the complexity of the modelling. Some studies have suggested that these base metals may have a negative effect on the purity and recovery of precious metals and should be considered for future refinement.

There are also practical considerations to consider that have not yet been fully discussed or cannot be included in the model at the current level of design. The variance or range of compositions of real-world e-waste streams may be

different from those modelled and using the technical model to simulate these ranges and identifying the system limits will be useful for evaluating the designs' feasibility. Additionally, investigating the benefits of process solution reuse rates over multiple loading cycles will enable a more refined understanding of the design and economics.

## 6. Conclusion

This paper focused on the design of an alternative small-scale, low-emissions process for the recovery of metals for e-waste that was economically feasible. Overall, the results of modelling have predicted that the proposed design was able to meet the design constraints and success criteria and is thus economically and environmentally feasible and likely to be profitable. The modelling predicted a carbon intensity under half of the current industrial route, while economic metrics suggested a payback period of under 2 years. Coupled with the potential social benefits of the current work, the design's achievement of metric targets provides a clear incentive for further investigation.

While overall findings are very positive, the modelling done cannot conclude that the proposed design would be feasible in practice. As has been discussed above, several assumptions were made to simplify the model into a solvable state. Additionally, there are many aspects of practical design and logistics that were not considered – for example, the ability to procure a steady supply of PCBs. If the process is to be developed further, it would be important to conduct more detailed environmental and economic analyses to support the design and validate the benefits of this processing method. Resolving the model limitations as discussed previously will be critical for improving the confidence in design feasibility, profitability, and sustainability, and will highlight routes for further refinement in this space. Validating the behavior of the system through lab and bench testing will be critical to verify expected performance and optimize the design for future process scale-up activities.

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