

Article

Modeling and Simulation of the Flotation Process in the Treatment of Critical Raw Materials

Jakub Kurty * , Michal Marcin , Martin Sisol , Jakub Progorowicz  and Ema Gánovská

Institute of Earth Resources, Faculty of Mining, Ecology, Process Control and Geotechnologies, Technical University of Košice, Letná 1/9, 042 00 Košice, Slovakia; michal.marcin@tuke.sk (M.M.); martin.sisol@tuke.sk (M.S.); ema.ganovska@tuke.sk (E.G.)

* Correspondence: jakub.kurty@tuke.sk; Tel.: +421-55-602-2992

Abstract

Flotation is one of the most widely used separation methods in mineral processing, and its performance is strongly influenced by complex physicochemical and hydrodynamic conditions. Flotation behaviour is commonly described using kinetic models. This study investigates the application of laboratory-derived flotation kinetic models for the simulation of a pilot-scale flotation circuit treating a polymetallic sulfide ore containing critical raw materials. Laboratory flotation experiments were conducted using Denver flotation cells, and kinetic parameters were determined by fitting first-order, two-component, and three-component kinetic models. The obtained parameters were implemented into an HSC SIM process model to evaluate the influence of flotation circuit configuration and residence time on process performance. The simulation indicated that the optimum circuit configuration consisted of two rougher and two scavenger flotation cells with an overall residence time of 28 min. The multi-component kinetic models provided a substantially better description of flotation behaviour than the classical first-order model and enabled a more realistic prediction of flotation performance. The simulation results were subsequently validated using a continuous pilot-scale flotation experiment. The simulated cleaner concentrate Cu grade (9.02 wt.%) was in good agreement with the experimental value (9.27 wt.%), confirming the capability of the proposed methodology to predict pilot-scale flotation performance. Although minor differences in product composition were observed, the overall flotation trends were reproduced successfully. The remaining discrepancies were mainly attributed to scale-up effects and differences in hydrodynamic conditions. The proposed methodology demonstrates that laboratory-derived kinetic parameters can be successfully applied to pilot-scale flotation process simulation and provides a practical tool for flotation circuit design and process optimization.

Keywords: flotation kinetics; kinetic modelling; process simulation; flotation simulation



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1. Introduction

Froth flotation is one of the most important physicochemical separation technologies for the selective recovery of valuable minerals from gangue materials. The process is fundamentally governed by differences in surface wettability, whereby hydrophobic particles preferentially attach to rising air bubbles and are subsequently transported into the froth phase, while hydrophilic particles remain dispersed in the pulp. Due to its high separation efficiency, operational flexibility, and economic viability, flotation is extensively

employed in mineral beneficiation, secondary resource utilization, wastewater treatment, and environmental remediation [1–7].

Flotation performance is strongly influenced by the complex interaction between mineralogical characteristics and hydrodynamic operating conditions. Parameters such as particle size distribution, surface chemistry, reagent type and dosage, pulp chemistry, aeration rate, and impeller speed directly affect the fundamental subprocesses of bubble–particle collision, attachment, and detachment, thereby determining flotation recovery and selectivity. Among these variables, particle size distribution is considered one of the most critical factors controlling flotation performance. Flotation recovery generally follows a characteristic size-dependent trend, with intermediate-sized particles exhibiting the highest recoveries, whereas both coarse and fine particles are recovered less efficiently. The reduced recovery of coarse particles is primarily attributed to their high settling velocities, which increase the likelihood of bubble–particle detachment and reduce froth recovery. In contrast, fine and ultrafine particles exhibit a low probability of colliding with and attaching to air bubbles because of their limited inertia and inability to overcome the fluid streamlines surrounding rising bubbles. Furthermore, fine particles are prone to non-selective recovery by entrainment with water into the froth, which can significantly reduce concentrate grade and flotation selectivity. Consequently, the particle size distribution of the feed not only governs flotation efficiency but also influences reagent consumption, froth stability, and the overall kinetics of the process. For this reason, flotation kinetics has become an essential quantitative tool for describing flotation behaviour, evaluating process performance, and optimizing operating conditions for improved recovery and selectivity [8–15].

Flotation kinetics describes the rate of mineral recovery as a function of flotation time and provides important insight into the dynamic mechanisms governing flotation systems. Among the various kinetic approaches, first-order kinetic models are most widely applied due to their simplicity and their ability to adequately describe many flotation systems. Early kinetic studies introduced the use of deterministic first-order rate constants to characterize flotation recovery, often including a maximum achievable recovery parameter. However, deviations from ideal first-order behaviour are frequently observed in practice, primarily due to the heterogeneous nature of mineral particles, which differ in size, liberation, and surface properties [16,17].

To address the limitations of conventional first-order kinetic models, more advanced approaches have been developed to account for the inherent heterogeneity of flotation systems. Among these, distributed first-order models describe flotation behavior through a continuous distribution of rate constants rather than a single value, accounting for the inherent heterogeneity of the system. This distribution reflects variations in particle size, liberation, and surface hydrophobicity, as well as differences in local hydrodynamic conditions such as bubble size and turbulence. By incorporating these effects, distributed models provide a more realistic representation of flotation kinetics and improve the description of recovery behavior in complex flotation systems. This framework recognizes that mineral particles do not exhibit uniform flotation responses but rather represent a spectrum of floatabilities. Building on this concept, flotation performance is increasingly interpreted as the combined response of multiple particle populations, each characterized by distinct kinetic parameters. Consequently, multi-component kinetic models have been introduced to explicitly represent fast- and slow-floating fractions within the flotation feed, typically associated with well-liberated, highly hydrophobic particles and composite or poorly hydrophobic particles, respectively. By distinguishing these fractions, multi-component models provide a more flexible and realistic description of non-ideal flotation kinetics compared to single-rate formulations. By accounting for the diverse flotation behavior of different mineral populations, these models provide a more realistic and accurate descrip-

tion of flotation kinetics and improve the prediction of recovery trends in complex mineral systems [18–23].

In addition to distributed and multi-component models, alternative kinetic formulations, including higher-order reaction models, have been proposed to describe deviations from first-order kinetics. However, it has been demonstrated that different model structures may lead to mathematically equivalent recovery expressions, which highlights the importance of careful model selection and interpretation when analysing flotation kinetics [24–27].

Kinetic analysis has been widely used to investigate the influence of mineralogical and operational variables on flotation performance. Statistical methods and machine-learning techniques have been integrated with flotation kinetic modelling to improve process prediction, parameter optimization, and intelligent process control. Furthermore, surface chemical properties, particularly the presence of hydrophilic oxygen-containing functional groups, have been reported to suppress flotation kinetics by decreasing mineral hydrophobicity and reducing the probability of stable bubble–particle attachment [8,28–33].

Despite these advances, there is still a lack of comprehensive studies that directly link laboratory-derived kinetic parameters with the simulation of pilot-scale flotation systems. In particular, the application of multi-component kinetic models for predicting the performance of larger-scale flotation circuits remains insufficiently explored.

Therefore, the aim of this study is to determine flotation kinetic parameters based on laboratory experiments, evaluate different kinetic models, and apply the obtained parameters in the simulation of a pilot-scale flotation process. The study focuses on assessing the suitability of various kinetic models and their impact on the prediction of flotation performance under different operating conditions.

2. Materials and Methods

2.1. Ore Sample

The material used in this study originated from the Zlaté Hory deposit and represents a predominantly sulfidic polymetallic ore containing copper, zinc, and iron. These elements are mainly present in the form of chalcopyrite (CuFeS_2), sphalerite (ZnS), and pyrite (FeS_2). The gangue fraction consists predominantly of oxide and silicate minerals, primarily quartz (SiO_2) and chamosite ($(\text{Fe}^{2+}, \text{Mg}, \text{Fe}^{3+})_5\text{Al}(\text{Si}_3\text{Al})\text{O}_{10}(\text{OH}, \text{O})_8$), which belongs to the chlorite group.

The raw feed material was milled for experimental testing. To ensure representativeness, the sample was homogenized and quartered using the coning and quartering technique. Representative subsamples were then obtained by riffle splitting for subsequent flotation and analytical procedures. All samples were stored in airtight polyethylene containers under dry, ambient conditions to prevent oxidation or contamination prior to testing.

2.2. Sample Characterization

The mineralogical composition of the sample was determined by powder X-ray diffraction (XRD) analysis using a PANalytical X'Pert PRO MPD diffractometer (Malvern Panalytical, Malvern, UK) equipped with a Co anode ($\lambda K\alpha = 0.17902 \text{ nm}$) and an X'Celerator RTMS detector (Malvern Panalytical, Malvern, UK), operating in conventional Bragg–Brentano reflection geometry. The measurements were performed over an angular range of $6\text{--}90^\circ 2\theta$ with a step size of $0.033^\circ 2\theta$ and a counting time of 240 s per step, resulting in a total scan time of approximately 1.5 h. The obtained diffraction data were processed using HighScore Plus 5.2 (Malvern Panalytical, Malvern, UK) and DIFFRAC plus TOPAS 4 (Bruker AXS,

Karlsruhe, Germany) software. Semi-quantitative phase analysis was performed using the Rietveld refinement method, and only crystalline phases were quantified.

The XRD results Figure 1 indicate that the sample is dominated by quartz (44.63%) and chlorite (29.38%), which represent the main gangue phases. Among sulphide minerals, pyrite (3.76%) and chalcopyrite (2.56%) were identified as the primary carriers of iron and copper, respectively. Minor amounts of albite (7.33%), ankerite (5.35%), muscovite (2.88%), and biotite (1.88%) were also detected.

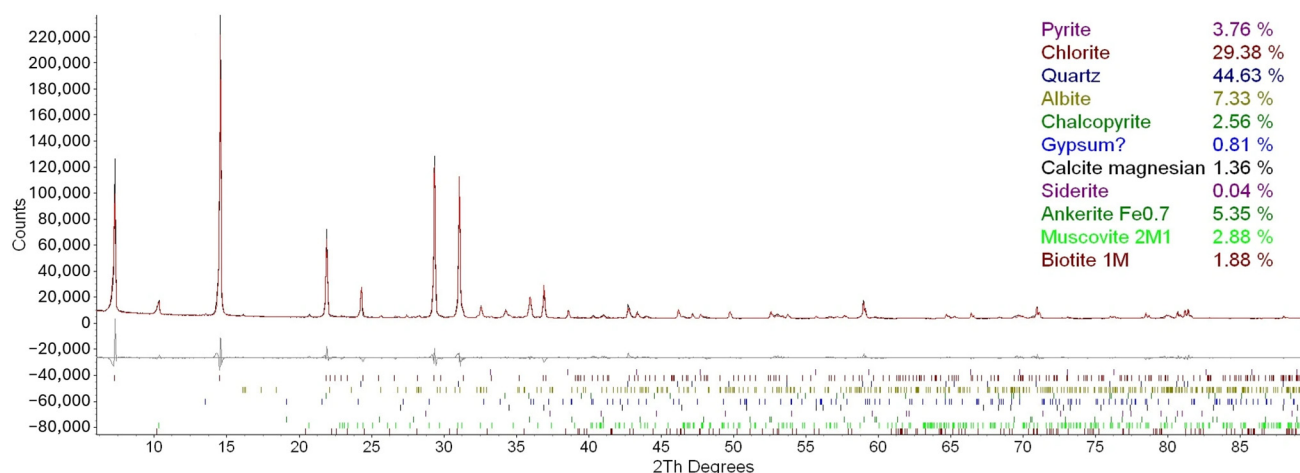


Figure 1. XRD pattern of the input material with identified mineral phases and their relative abundances.

The particle size distribution “Mastersizer analysis” of the ore sample was analyzed using a Malvern Mastersizer 3000 laser diffraction instrument (Malvern Instruments Ltd., Malvern, UK). A representative subsample was prepared by dispersing the material in demineralized water to form a slurry with a solids concentration of approximately 30 wt.%, maintained at ambient temperature. To ensure thorough dispersion of the mineral particles, the suspension was stirred at 690 rpm for 10 min. Each subsample was analyzed three times, and the final particle size distribution was calculated as the average of 15 in-dividual measurements.

Quantitative elemental analysis was performed using X-ray fluorescence (XRF) (SPECTRO Analytical Instruments GmbH, Kleve, Germany) and atomic absorption spectroscopy (AAS) (iCE 3300 AA, Thermo Fisher Scientific, Grand Island, NY, USA), depending on the target elements and required detection limits. The chemical analysis of the raw ore is shown in Table 1.

Table 1. Chemical composition of the input material.

Element	Average Content (wt.%)
Cu	0.85
Zn	0.14
Fe	11.8
S	2.36

2.3. Laboratory Flotation Experiments

Laboratory flotation experiments were conducted using an automated froth flotation unit equipped with 2 L and 14 L Denver-type cells (Outotec GTK LabCell, Metso, Espoo, Finland). All experiments were performed at an ambient temperature of 25 °C. The natural pulp pH was not adjusted and remained in the range of 7.5–7.8 throughout the experiments. The slurry density was maintained at 450 g·L^{−1} during the rougher and scavenger flotation stages and 200 g·L^{−1} during the cleaner flotation stage. Pre-weighed

samples were introduced into the flotation cell and mixed with water to form a slurry. Agitation was initiated at 1000 rpm, followed by the addition of the collector and frother. The slurry was conditioned for 7 min, after which air was introduced at a constant flow rate of 4 NL·min⁻¹. Froth products from the rougher stage were collected at time intervals of 3, 6, 9, 12, and 15 min, while froth products from the cleaner stage were collected at time intervals of 4, 8, and 12 min.

After completion of flotation, the concentrates were collected, dried, and analyzed using X-ray fluorescence (XRF). Metal recoveries for each concentrate were calculated according to Equation (1).

$$\varepsilon = \frac{C_c}{F_f} \times 100\% \quad (1)$$

where C and F represent the masses of the flotation concentrate and feed, respectively, and c and f denote the corresponding metal grades in the concentrate and feed.

The laboratory flotation test flowsheets for individual stages are presented in Figures 2–4.

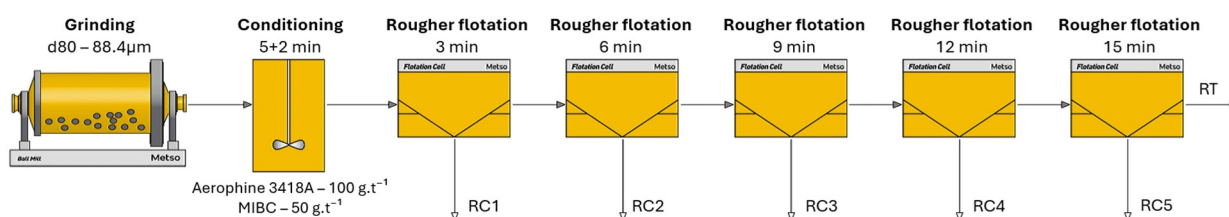


Figure 2. Laboratory rougher flotation test flowsheet. The abbreviations for the test products shown in the flowsheet are RC1–5 for rougher concentrates 1–5 and RT for rougher tailings.

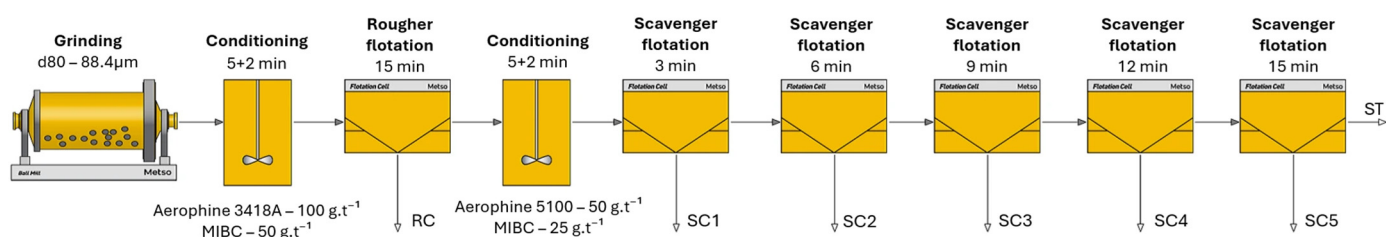


Figure 3. Laboratory scavenger flotation test flowsheet. The abbreviations for the test products shown in the flowsheet are RC for rougher concentrate, SC1–5 for scavenger concentrates 1–5 and ST for scavenger tailings.

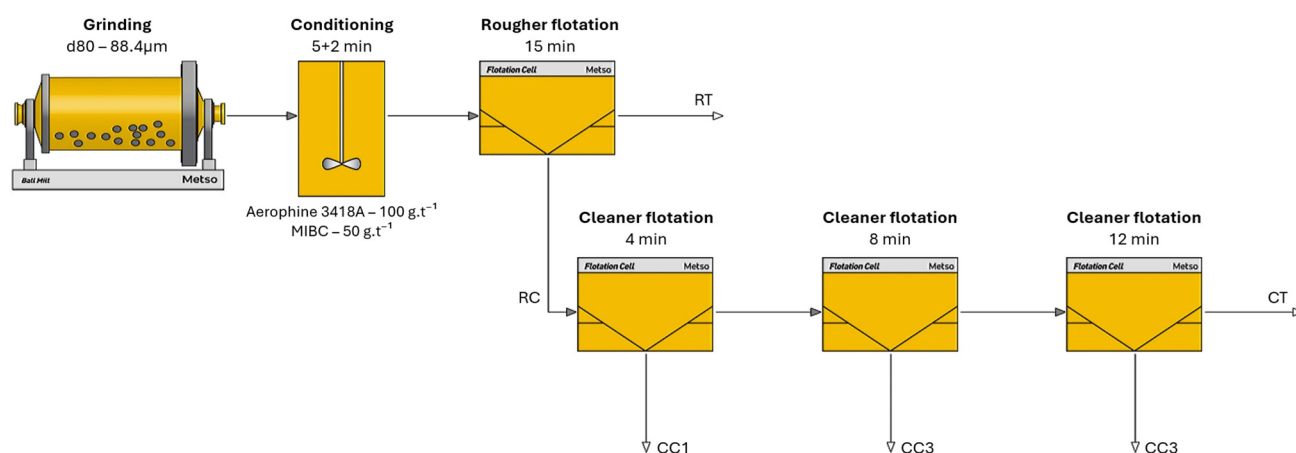


Figure 4. Laboratory rougher-cleaner flotation test flowsheet. The abbreviations for the test products shown in the flowsheet are RC for rougher concentrate, CC1–3 for scavenger concentrates 1–3 and RT and CC for rougher and cleaner tailings.

All reagents used for Froth flotation tests are summarized in Table 2.

Table 2. Reagents used for froth flotation tests. MIBC—Methyl isobutyl carbinol.

Reagent	Dosage [g·t ^{−1}]	Purity	Purpose	Producer	City/State
Aerophine 3418A	100	Analytical	Collector	Cytec Industries Inc.	Princeton, NJ, USA
Aerophine 5100	50	Analytical	Collector	Cytec Industries Inc.	Princeton, NJ, USA
MIBC	50/25	Analytical	Frother	Thermo Fisher Scientific	Ward Hill, MA, USA

2.4. Modelling Kinetic Parameters

Calculated metal recoveries obtained from laboratory flotation experiments were used for the modelling of kinetic parameters. The HSC SIM module of HSC Chemistry[®] software (version 10.8.1.2) was employed for this purpose. Mineral processing models in HSC SIM are based on mineral recoveries derived from assayed elemental compositions. To ensure consistency of the experimental data, assay measurements were reconciled using the HSC Mass Balance module for each flotation test. The reconciled assay data were subsequently converted into mineral phases using the HSC Geo module. A simplified mineralogical composition was applied during this conversion: chalcopyrite (Ccp) represented copper sulfides, sphalerite (Sp) represented zinc sulfides, pyrite (Py) represented the remaining sulfide minerals, and quartz (Qtz) represented non-floating gangue minerals.

Following data reconciliation and mineral conversion, kinetic parameters were fitted to the laboratory flotation recoveries using cumulative residence times (τ) of 5, 10, and 15 min for the concentrate fractions. The fitting procedure was performed using the HSC Model Fit module, applying multiple kinetic models: first-order, two-component Kelsall model, and three-component Kelsall model [34].

The batch flotation kinetics were described using the following equations:

$$R = R_{\infty} (1 - e^{-kt}) \quad (2)$$

where k is the flotation rate constant and R_{∞} is the ultimate recovery [26,34].

$$R = 100 \times \left[(1 - \phi) (1 - e^{-k_f t}) + \phi (1 - e^{-k_s t}) \right] \quad (3)$$

where ϕ represents the fraction of particles with the slow flotation rate constant k_s , and k_f is the fast flotation rate constant [26,34]. This model assumes a theoretical ultimate recovery of 100%.

$$R = 100 \times \left[\phi_f (1 - e^{-k_f t}) + \phi_s (1 - e^{-k_s t}) + \phi_n (0) \right] \quad (4)$$

$$\phi_f + \phi_s + \phi_n = 1 \quad (5)$$

where ϕ_f , ϕ_s , and ϕ_n represent the fractions of fast-floating, slow-floating, and non-floating particles, respectively, and k_f and k_s are the corresponding flotation rate constants [26,34].

The obtained kinetic parameters were subsequently used as input for the simulation of a pilot-scale flotation process, with the aim of enabling comparison between simulated and experimental results.

2.5. Simulation of Pilot-Scale Flotation

Simulation was carried out using the HSC SIM module of HSC Chemistry[®] software. For this purpose, mineral processing unit models, namely the Conditioner and Flotation Cell, were applied. The Conditioner model was coupled with flotation models simulating the separation between floating and non-floating particles based on the selected operating

mode and residence time. A simplified flowsheet of the simulated flotation circuit is shown in Figure 5.

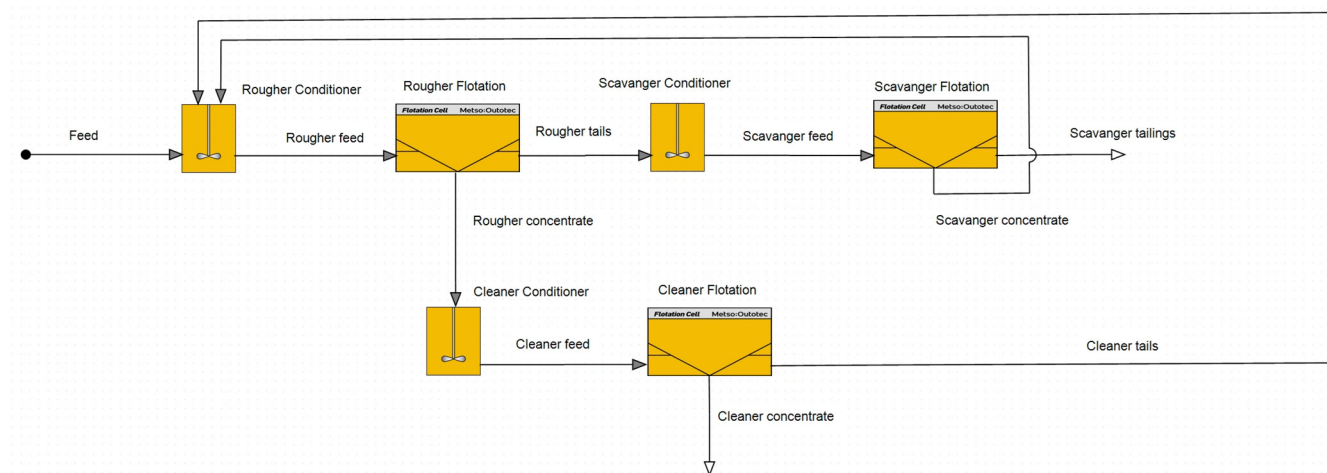


Figure 5. Simplified flowsheet of the simulated pilot-scale flotation circuit used in HSC SIM, including rougher, scavenger, and cleaner stages.

The simulation was performed under continuous operation with a feed rate of $50 \text{ kg} \cdot \text{h}^{-1}$, representing typical pilot-scale conditions and enabling a realistic assessment of flotation circuit performance.

Froth recovery and entrainment were not considered in the simulations; therefore, only true flotation (i.e., recovery from the pulp zone) was taken into account. During the simulations, kinetic parameters were applied to the corresponding flotation stages. All simulations were conducted under steady-state conditions, assuming ideal mixing within each flotation unit.

This simulation approach enables the evaluation of flotation circuit performance under varying operational conditions without the need for extensive pilot-scale testing. However, it should be noted that the neglect of entrainment and froth recovery may lead to deviations from real flotation performance, particularly for fine particles.

2.6. Pilot-Scale Flotation

A pilot-scale flotation experiment was conducted to validate the laboratory-derived kinetic model under continuous operating conditions. The pilot flotation circuit consisted of 50 L Denver-type flotation cells used for the rougher and scavenger stages and a 17 L Denver-type flotation cell used for the cleaner stage. The flotation cells were designed and manufactured by the Prototyping and Innovation Centre, Faculty of Mechanical Engineering, Technical University of Košice (Slovakia).

The 50 L flotation unit consisted of two interconnected 25 L chambers, each equipped with an independent rotor of 85 mm diameter. The cleaner flotation stage was performed in a single 17 L flotation cell equipped with an 85 mm diameter rotor, providing hydrodynamic conditions comparable to those of the rougher and scavenger cells.

Both flotation cell designs incorporated an adjustable overflow weir, allowing continuous adjustment of the pulp level and froth height during operation. Material transport between flotation cells was achieved using peristaltic pumps, ensuring a constant slurry flow independent of slurry density and solids content while minimizing wear caused by abrasive particles.

The flotation circuit was operated at a slurry density of $450 \text{ g} \cdot \text{L}^{-1}$ in the feed to rougher and scavenger stages. The rougher and scavenger cells were operated at an impeller speed of 1200 rpm with an air flow rate of $6 \text{ NL} \cdot \text{min}^{-1}$, whereas the cleaner cell was operated at

1200 rpm and $5 \text{ NL} \cdot \text{min}^{-1}$. The residence time in the rougher and scavenger stages was set to 28 min, based on the simulation results obtained from the HSC SIM model.

The pilot-scale flotation experiment was divided into three consecutive stages: start-up and stabilization, steady-state operation, and shutdown.

During the start-up and stabilization stage, the flotation circuit, including the conditioning tank and flotation cells, was gradually filled with slurry. Froth concentrate produced during this period was continuously transferred to the cleaner stage and collected separately as the “<0 min” product. Once the circuit was filled, the operating parameters, primarily pulp level and air flow rate, were adjusted to achieve stable operating conditions.

After stabilization, the experiment proceeded to the steady-state operation stage. During this period, concentrate and tailings samples were collected at 60 min intervals, corresponding to the time intervals 0–60 min, 60–120 min, and 120–180 min. Only minor adjustments of the operating parameters were performed to maintain stable process conditions throughout the experiment.

Following the processing of the entire slurry batch, the experiment entered the shutdown stage. During this period, the flotation circuit was gradually emptied while the froth concentrate continued to be transferred to the cleaner stage until froth production in the rougher stage ceased. Products collected during this period were labelled as the “>180 min” fraction.

All concentrate and tailings samples were subsequently dried, weighed, and homogenized. Representative subsamples were prepared for chemical analysis, and the elemental composition was determined by X-ray fluorescence (XRF). Each sample was analyzed several times, and the reported values represent the average of the individual measurements.

3. Results and Discussion

The results of laboratory flotation experiments, kinetic modelling, and subsequent process simulation are presented and discussed in this section. The analysis focuses on the evaluation of flotation kinetics, the comparison of different kinetic models, and the application of the obtained parameters in the simulation of a pilot-scale flotation circuit.

3.1. Grain Size Analysis

The particle size distribution (PSD) of the ore sample, obtained by “Mastersizer analysis”, is represented in Figure 6. According to the laser scattering data, the particle size distribution of the sample ranged from 0.1 to 1000 μm , with a P_{80} of 88.4 μm .

3.2. Laboratory Flotation Results

The results of laboratory flotation experiments are presented as recovery–time profiles for the investigated minerals presented in Figures 6 and 7. The flotation kinetics exhibited a typical trend, with rapid initial recovery followed by a gradual decrease in flotation rate over time. This behaviour indicates the presence of particle populations with different flotation responses, where fast-floating particles are recovered in the initial stage, while slower-floating particles contribute to recovery at later times.

The experimental data obtained from laboratory flotation tests were used as input for mass balance calculations and subsequent kinetic modelling. The assay data for individual flotation products were reconciled using the HSC Mass Balance module to ensure consistency of the dataset prior to kinetic parameter estimation.

A summary of the experimental results is presented in Table 3. These data represent the basis for the calculation of mineral recoveries and the subsequent fitting of kinetic models. The use of mass-balanced data ensures that the derived kinetic parameters are physically consistent and suitable for process simulation.

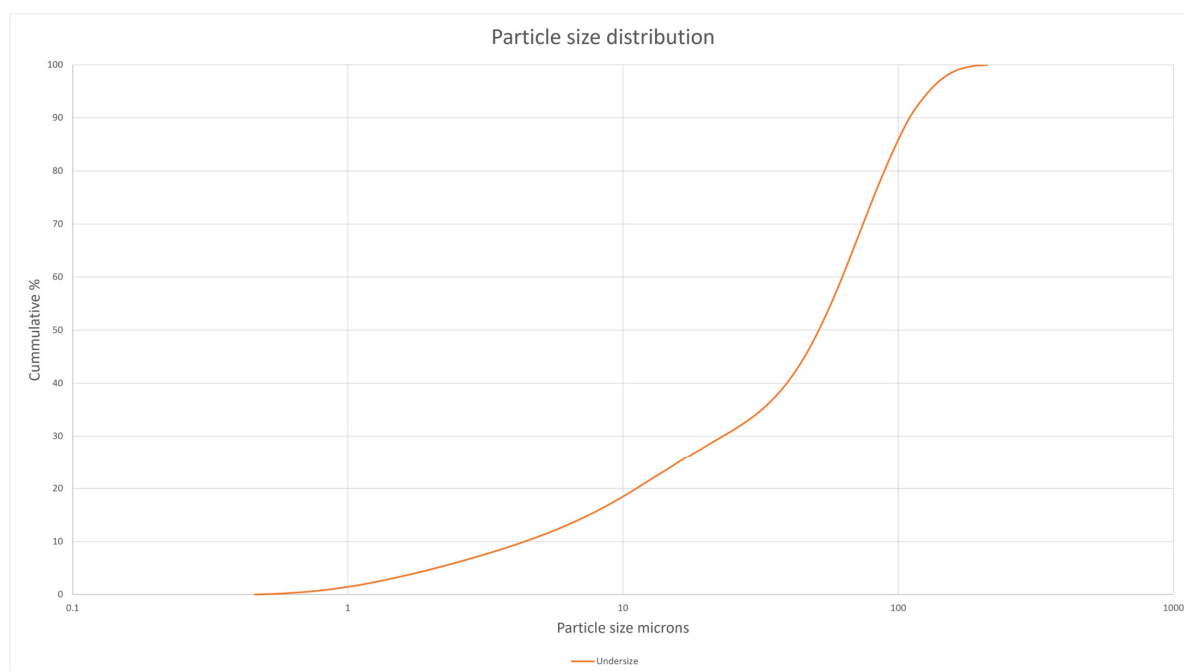


Figure 6. Particle size distribution of sample.

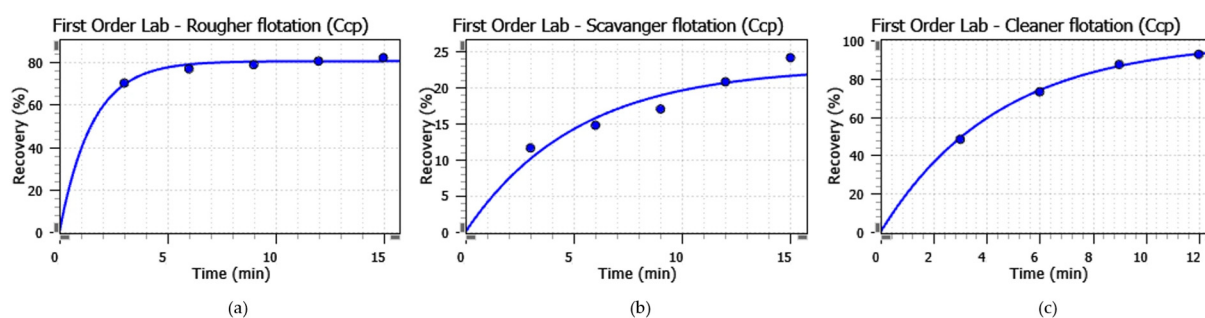


Figure 7. First-order kinetic model fit for chalcopyrite (Ccp) in (a) rougher, (b) scavenger, and (c) cleaner flotation stages.

Table 3. Summary of laboratory flotation results used for kinetic modelling.

Stream	Mass (g)	wt Cu (%)	wt Zn (%)	wt S (%)
Feed	5351	0.792	0.1570	3.93
RC1	378	7.87	1.342	31.62
RC2	98.2	2.888	0.436	12.05
RC3	36.3	2.344	0.352	11.81
RC4	42.8	1.629	0.2462	10.70
RC5	32.1	2.131	0.324	7.84
SC1	121.3	0.722	0.1104	32.2
SC2	31.60	0.746	0.1048	11.57
SC3	20.70	0.822	0.1228	10.34
SC4	23.60	1.199	0.2009	5.93
SC5	21.20	1.187	0.1960	5.41
Cleaner feed	3585	0.792	0.1570	3.93
CC1	106.4	10.77	1.787	38.8
CC2	68.2	9.70	1.542	39.7
CC3	57.2	6.89	0.998	40.1
CC4	21.46	6.30	0.916	36.8

3.3. Kinetic Modelling Results

The experimental flotation data were fitted using different kinetic models, including first-order, two-component, and three-component models. The quality of the fit was evaluated using the coefficient of determination (R^2). The results show that multi-component models provided a significantly better fit to the experimental data compared to the first-order model. This is attributed to the ability of multi-component models to account for the heterogeneous nature of mineral particles, which exhibit different flotation rate constants depending on their surface properties and degree of liberation. The first-order model showed limited agreement with the experimental data, particularly in the later stages of flotation, and was therefore not suitable for describing the full kinetic behaviour of the system.

The resulting kinetic parameters for the rougher, scavenger and cleaner flotation stages are presented in Table 4, Table 5, Table 6 respectively.

Table 4. Kinetic parameters of Rougher flotation.

Kinetic Model	Parameter	Ccp	Sp	Py	Qtz
First order	k	0.662	0.712	0.616	0.181
	R_∞	80.501	68.004	62.3	5.872
	R^2	0.99	0.991	0.981	0.987
Two fraction Kellsal model	$\emptyset$	0.269	0.37	0.451	0.959
	k_f	0.892	0.92	0.898	0.255
	k_s	0.027	0.013	0.016	0.001
	R^2	1	1	1	1
Three fraction Kellsal model	$\emptyset_f$	0.731	0.629	0.521	0.027
	$\emptyset_s$	0.269	0.355	0.176	0.168
	$\emptyset_n$	0	0.015	0.302	0.805
	k_f	0.892	0.921	1.012	0.42
	k_s	0.027	0.014	0.077	0.014
	R^2	1	1	1	0.994

Table 5. Kinetic parameters of Scavenger flotation.

Kinetic Model	Parameter	Ccp	Sp	Py	Qtz
Two fraction Kellsal model	$\emptyset$	0.918	0.963	0.337	0.983
	k_f	2.18	1.44	0.928	0.191
	k_s	0.012	0.005	0.027	0.001
	R^2	0.994	0.989	0.999	0.984
Three fraction Kellsal model	$\emptyset_f$	0.082	0.037	0.584	0.008
	$\emptyset_s$	0.918	0.657	0.215	0.205
	$\emptyset_n$	0	0.306	0.201	0.787
	k_f	2.176	1.493	1.366	0.51
	k_s	0.012	0.007	0.143	0.007
	R^2	0.994	0.998	1	0.998

The flotation kinetics of chalcopyrite (Ccp) for the rougher, scavenger, and cleaner flotation stages are presented in Figure 7. The experimental data were fitted using the first-order kinetic model to evaluate its suitability for describing the flotation behaviour. In the rougher flotation stage Figure 7a, a rapid increase in recovery was observed during the initial flotation period, followed by a gradual approach to a plateau value. This trend indicates the presence of fast-floating particles that are recovered at early stages of flotation. The first-order model provided a reasonable fit for the rougher stage, particularly in the

initial region of the curve. In contrast, the scavenger flotation stage Figure 7b exhibited a slower increase in recovery over time, reflecting the presence of particles with lower floatability. The deviation between experimental data and the fitted curve suggests that the first-order model does not fully capture the flotation behaviour in this stage. For the cleaner flotation stage Figure 7c, the recovery increased more gradually compared to the rougher stage, with a smoother transition toward the final recovery value. Although the first-order model was able to describe the general trend, discrepancies between the model and experimental data indicate the presence of particle populations with different flotation rates.

Table 6. Kinetic parameters of Cleaner flotation.

Kinetic Model	Parameter	Ccp	Sp	Py	Qtz
Two fraction Kellsal model	$\emptyset$	0.005	0.046	1	0.910
	k_f	0.227	0.225	0.221	0.376
	k_s	0	0	0.207	0.020
	R^2	0.999	0.998	0.922	0.996
Three fraction Kellsal model	$\emptyset_f$	0.995	0.954	0	0.029
	$\emptyset_s$	0	0	1	0.429
	$\emptyset_n$	0.005	0.046	0	0.542
	k_f	0.227	0.225	0.221	1.639
	k_s	0.225	0.224	0.207	0.073
	R^2	0.999	0.998	0.922	0.998

Overall, the results demonstrate that the first-order kinetic model is insufficient to fully describe the flotation kinetics across all flotation stages. This limitation is particularly evident in the scavenger stage, where particle heterogeneity plays a more significant role. Consequently, multi-component kinetic models are required to accurately represent the flotation behavior of the system.

The flotation kinetics of chalcopyrite (Ccp) were further evaluated using the two-component kinetic model, as shown in Figure 8. In comparison with the first-order model, a significantly improved agreement between the experimental data and the fitted curves was observed across all flotation stages. In the rougher flotation stage Figure 8a, the two-component model accurately describes both the rapid initial increase in recovery and the subsequent gradual approach to the final recovery. This indicates that the model effectively captures the presence of both fast-floating and slow-floating particle fractions. A similar improvement was observed in the scavenger stage Figure 8b, where the two-component model better represents the gradual increase in recovery over time. The improved fit suggests that the flotation behaviour in this stage is strongly influenced by slow-floating particles, which are not adequately described by a single rate constant. In the cleaner flotation stage Figure 8c, the two-component model also provided a good agreement with the experimental data, particularly in the intermediate and final stages of flotation. This confirms the presence of particle populations with different flotation kinetics even in the cleaner stage.

Overall, the results demonstrate that the two-component kinetic model provides a more accurate and physically meaningful description of flotation kinetics compared to the first-order model. The improved fit is attributed to the model's ability to account for particle heterogeneity, which is a fundamental characteristic of real flotation systems.

The flotation kinetics were also evaluated using the three-component kinetic model. However, only marginal differences were observed between the two-component and three-component models in terms of fitting accuracy.

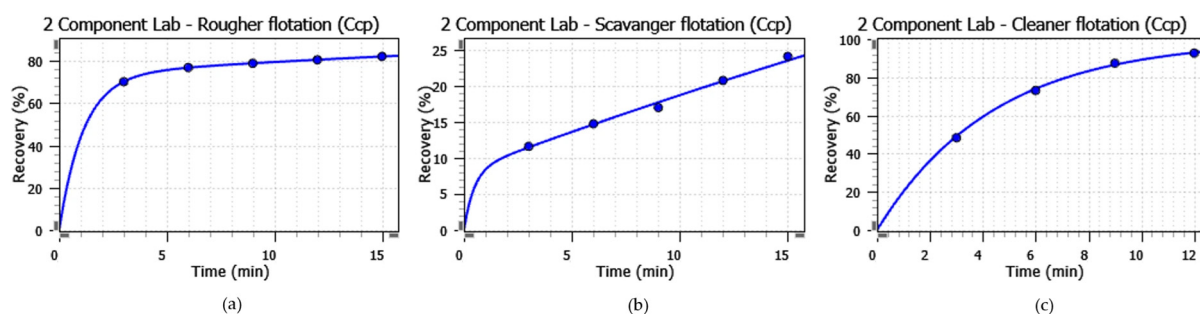


Figure 8. Two-component kinetic model fit for chalcopyrite (Ccp) in (a) rougher, (b) scavenger, and (c) cleaner flotation stages.

A comparison of the coefficient of determination (R^2) values for different kinetic models is presented in Table 7. The results clearly show that multi-component models provide a significantly better fit to the experimental data compared to the first-order model.

Table 7. Comparison of the coefficient of determination (R^2) for different kinetic models.

Flotation Stage	First Order				Two Fraction				Three Fraction			
	Ccp	Sp	Py	Qtz	Ccp	Sp	Py	Qtz	Ccp	Sp	Py	Qtz
Rougher	0.99	0.991	0.981	0.987	1	1	1	0.995	1	1	1	0.994
Scavenger	0.906	0.885	0.982	0.464	0.994	0.989	0.999	0.984	0.994	0.988	1	0.998
Cleaner	0.999	0.998	0.922	0.996	0.999	0.998	0.922	0.996	0.999	0.998	0.922	0.998

In the rougher flotation stage, the first-order model already provides a relatively good fit; however, the multi-component models achieve near-perfect agreement with the experimental data. In contrast, in the scavenger stage, the first-order model exhibits noticeably lower R^2 values, particularly for quartz, indicating its limited ability to describe the flotation behaviour of less floatable particles.

Both the two-component and three-component models show consistently high R^2 values across all flotation stages and minerals, demonstrating their suitability for describing flotation kinetics in heterogeneous systems. However, only marginal differences were observed between the two-component and three-component models.

This suggests that the two-component model is sufficient to accurately describe the flotation kinetics while maintaining a simpler model structure. The limited improvement obtained with the three-component model indicates that the flotation behaviour of the investigated system can be effectively represented by two dominant particle populations. The lowest R^2 values observed for the first-order model in the scavenger stage highlight the importance of accounting for slow-floating particle fractions.

The rougher stage was adequately described by the two-component model, indicating that the flotation process was dominated by fast- and slow-floating particle populations. In contrast, the cleaner and scavenger stages were better represented by the three-component model, suggesting the presence of an additional non-floating particle fraction. This observation reflects the progressive depletion of readily floatable particles throughout the flotation circuit and demonstrates the increasing heterogeneity of particle flotation behaviour in the downstream stages.

3.4. Simulation of Pilot Plant

The selection of kinetic models used in the simulation was directly based on the results of laboratory kinetic modelling. As demonstrated by the comparison of R^2 values, multi-component models provided a significantly better fit to experimental data compared

to the first-order model. In particular, the two-component model was identified as the most suitable compromise between model accuracy and complexity. Therefore, the kinetic parameters derived from the two-component model were subsequently used as input for the simulation of the flotation circuit.

The improved representation of particle heterogeneity in the two-component model allowed for a more realistic prediction of flotation performance, especially in stages where slow-floating particles significantly influence overall recovery.

The simulation results provided indicative values of mineral distribution across flotation products, as shown in Table 8. The results demonstrate that chalcopyrite (Ccp) is predominantly concentrated in the final concentrate, while quartz (Qtz) is primarily distributed to the tailings. Elemental compositions obtained from the simulation were converted into mineral phases based on the simplified mineralogical model described in Section 2.4.

Table 8. Simulated mineral composition and mass distribution of flotation products.

Product	Mass (kg)	Yield (%)	Ccp	Sp	Py	Qtz
Cleaner concentrate	3.96	7.92	21.78	1.75	63.52	12.95
Scavenger tailings	46.84	93.68	0.64	0.11	0.83	98.42

The simulation results indicate that chalcopyrite is effectively concentrated in the final concentrate, confirming the selectivity of the flotation process. In contrast, quartz is predominantly rejected to the tailings, demonstrating effective gangue separation. These trends are consistent with the expected flotation behavior of sulfide and gangue minerals.

3.5. Pilot-Scale Verification of Simulation Model

To verify the predictive capability of the proposed simulation model, a continuous pilot-scale flotation experiment was conducted under the operating conditions determined by the simulation. The pilot-scale experiment provided experimental data for validating the simulation model under continuous operating conditions and evaluating its applicability for process design and optimization.

The results of the pilot-scale flotation experiment are first presented in Table 9, followed by a comparison between the simulated and experimental results. The comparison was performed using the final products of the proposed flotation circuit, namely the cleaner concentrate and the scavenger tailings. Both mineralogical (XRD) and chemical (XRF) analyses were used for the evaluation.

Table 9. Results of pilot-scale flotation experiment.

Product	Sampling Interval (min)	Yield (%)	Cu Grade (%)	S Grade (%)
Concentrate	<0	-	9.64	19.88
	0–60	4.5	8.14	21.71
	60–120	9.9	7.87	21.48
	120–180	12.0	10.68	36.08
	>180	13.9	8.34	44.68
Tailings	0–60	95.5	0.17	0.85
	60–120	90.1	0.14	0.9
	120–180	88.0	0.14	0.92
	>180	86.1	0.12	0.68

Since the simulation model considered only four mineral phases—chalcopyrite, sphalerite, pyrite, and quartz—where quartz represented all remaining gangue minerals, the XRD results were processed to ensure consistency with the simulation model. The contents

of all identified gangue minerals, excluding chalcopyrite, sphalerite, and pyrite, were summed and represented as quartz. This simplification enabled a direct comparison between the experimental mineralogical composition and the mineral fractions defined in the simulation model.

Because XRD provides semi-quantitative mineralogical information, the comparison was complemented by quantitative XRF analysis, allowing a more reliable assessment of the agreement between the simulated and experimental results.

The results of the pilot-scale flotation experiment are summarized in Table 9. Following the start-up period, the flotation circuit reached stable operating conditions, and samples of the concentrate and tailings were collected at 60 min intervals during steady-state operation. The concentrate mass yield gradually increased throughout the experiment, while the copper grade remained within the range of 7.87–10.68 wt.% during steady-state operation. The sulfur content in the final tailings remained below 1 wt.% in all sampling intervals, indicating that the target sulfur level in the tailings was successfully achieved. To eliminate the influence of the start-up and shutdown periods, only the samples collected during steady-state operation (60–120 and 120–180 min) were used for the subsequent comparison with the simulation results.

For the comparison between the experimental XRD results and the simulation output, the mineralogical data were adjusted to match the simplified mineral set used in the HSC SIM model. The simulation model considered four mineral phases: chalcopyrite (Ccp), sphalerite (Sp), pyrite (Py), and quartz (Qtz). In this simplified representation, quartz was used as a proxy for the non-floating gangue fraction. Therefore, the experimental quartz value reported in the comparison tables was calculated as the sum of all gangue mineral phases identified by XRD, excluding chalcopyrite, sphalerite, and pyrite. In this context, the term “quartz” in the comparison tables does not represent pure quartz only, but the total gangue fraction expressed as Qtz in the simulation model. This approach allowed direct comparison between the experimentally determined mineralogical composition and the mineral phases defined in the simulation model.

The comparison between the simulated and experimental cleaner concentrate is presented in Table 10. In contrast to the laboratory-scale validation, larger deviations between the simulation and the pilot-scale experiment were observed. These differences are primarily attributed to scale-up effects resulting from changes in the hydrodynamic conditions associated with the increased flotation cell volume.

Table 10. Comparison of simulated and experimental cleaner concentrate composition.

Parameter	Pilot-Scale (60–120)	Pilot-Scale (120–180)	Average	Simulation
Ccp (%)	35.6	27.0	31.3	21.78
Sp (%)	3.0	2.3	2.65	1.75
Py (%)	54.7	61.3	58.0	63.52
Qtz (%)	6.9	9.5	8.2	12.95

The simulation underestimated the chalcopyrite content while overestimating the quartz content in the cleaner concentrate. Nevertheless, the agreement between the simulated and experimentally determined chemical composition was considerably better. The simulated Cu grade (9.02 wt.%) was in close agreement with the average experimental value (9.27 wt.%), corresponding to a relative deviation of approximately 3%.

Although noticeable differences were observed between the individual sampling intervals (60–120 and 120–180 min), the average composition showed good agreement with the simulation results. This indicates that the simulation model is capable of predicting

the overall concentrate quality despite natural fluctuations occurring during continuous pilot-scale operation.

The comparison of the scavenger tailings is presented in Table 11. Similar to the cleaner concentrate, the simulation predicted lower contents of valuable minerals in the tailings than those measured experimentally. Consequently, the model slightly underestimated the sulfur content and overestimated the gangue mineral content.

Table 11. Comparison of simulated and experimental scavenger tailings composition.

Parameter	Pilot-Scale (60–120)	Pilot-Scale (120–180)	Average	Simulation
Ccp (%)	1.2	1.2	1.2	0.64
Sp (%)	0	0	0	0.11
Py (%)	1.5	1.4	1.45	0.83
Qtz (%)	97.3	97.3	97.3	98.42

Despite these discrepancies, the simulation correctly predicted the overall flotation trend and the achievement of the target sulfur content below 1 wt.% in the final tailings. The observed deviations are considered a consequence of scale-up effects, including differences in mixing intensity, bubble size distribution, and particle–bubble collision probability between the laboratory and pilot-scale flotation cells.

Overall, the comparison demonstrates that the simulation model provides a reliable prediction of the flotation circuit performance and is suitable for the design and optimization of pilot-scale flotation circuits. Future work should focus on incorporating scale-up correction factors to further improve the quantitative agreement between simulated and experimental results.

4. Conclusions

This study presented a workflow for the development and validation of a flotation process simulation model based on laboratory-scale flotation experiments, kinetic modelling, and pilot-scale verification. Laboratory flotation data were used to determine flotation kinetic parameters, which were subsequently implemented in the HSC SIM environment to simulate the performance of a pilot-scale flotation circuit.

The comparison of different kinetic models demonstrated that the two-component model provided the most suitable description of flotation behaviour in the rougher stage, whereas the three-component model more accurately represented the cleaner and scavenger stages. Compared with the classical first-order model, the multi-component kinetic models significantly improved the agreement between the fitted and experimental flotation kinetics, providing a more realistic representation of heterogeneous particle populations. This observation is consistent with previous studies, which demonstrated that multi-component kinetic models provide a more realistic description of heterogeneous particle populations than classical first-order models by accounting for fast- and slow-floating fractions of valuable minerals [18,35].

The developed simulation model was successfully applied to evaluate the influence of key operating parameters, including residence time and flotation circuit configuration. The simulation results identified an optimum residence time of 28 min and a flotation circuit consisting of two rougher and two scavenger flotation cells for the investigated ore.

The pilot-scale flotation experiment confirmed that the proposed simulation methodology correctly predicted the overall behaviour of the flotation circuit. Although differences between the simulated and experimental results were observed, mainly due to scale-up effects and changes in hydrodynamic conditions, the simulation accurately reproduced the overall trends in concentrate quality and tailings composition. The comparison of XRD

and XRF analyses demonstrated good agreement between the simulated and experimental products, confirming the applicability of the proposed modelling approach for pilot-scale flotation process design. The observed differences between the laboratory-based simulation and the pilot-scale experiment confirm previous findings that flotation scale-up remains strongly dependent on hydrodynamic conditions and cell-specific operating characteristics. Nevertheless, the good agreement between the predicted and experimental concentrate grades demonstrates that laboratory-derived kinetic parameters can provide a reliable basis for flotation circuit simulation and optimization [36].

The proposed methodology provides a practical framework for integrating laboratory flotation experiments, kinetic modelling, process simulation, and pilot-scale verification into a single workflow. Such an approach can significantly reduce the number of pilot-scale experiments required during process development and represents a useful tool for the design and optimization of flotation circuits treating critical raw materials.

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References

1. Park, H.; Wang, L. Experimental Studies and Modeling of Surface Bubble Behaviour in Froth Flotation. *Chem. Eng. Res. Des.* **2015**, *101*, 98–106. [CrossRef]
2. Woodburn, E.T. Physico-Chemical Elementary Processes in Flotation. By H. J. SCHULZE. Elsevier, 1984. 320 pp. Dfl 175.00. *J. Fluid. Mech.* **1985**, *152*, 501–502. [CrossRef]
3. Falagán, C.; Grail, B.M.; Johnson, D.B. New Approaches for Extracting and Recovering Metals from Mine Tailings. *Miner. Eng.* **2017**, *106*, 71–78. [CrossRef]
4. Ri, K.; Ri, J.; Kim, K.; Ri, H.; Song, R.; Kim, C. Application of Flotation on the Recovery/Removal of Heavy Metals in Secondary Resources: A Comprehensive Review. *Miner. Eng.* **2024**, *218*, 108999. [CrossRef]
5. Kyzas, G.Z.; Matis, K.A. Flotation in Water and Wastewater Treatment. *Processes* **2018**, *6*, 116. [CrossRef]
6. Rubio, J.; Souza, M.L.; Smith, R.W. Overview of Flotation as a Wastewater Treatment Technique. *Miner. Eng.* **2002**, *15*, 139–155. [CrossRef]
7. Nguyen, A. *Schulze Colloidal Science of Flotation*; CRC Press: Boca Raton, FL, USA, 2004; ISBN 0-8247-4782-8.
8. Wang, X.; Zhang, Q. Role of Surface Roughness in the Wettability, Surface Energy and Flotation Kinetics of Calcite. *Powder Technol.* **2020**, *371*, 55–63. [CrossRef]
9. Sokolović, J.; Miskovic, S. The Effect of Particle Size on Coal Flotation Kinetics: A Review. *Physicochem. Probl. Mineral. Process.* **2018**, *54*, 1172–1190. [CrossRef]
10. Newcombe, B.; Akerstrom, B.; Jaques, E. The Effect of Rotor Speed on the Flash Flotation Performance of Au and Cu in an Industrial Concentrator. *Miner. Eng.* **2018**, *124*, 28–43. [CrossRef]
11. Polat, M.; Polat, H.; Chander, S. Physical and Chemical Interactions in Coal Flotation. *Int. J. Miner. Process.* **2003**, *72*, 199–213. [CrossRef]

12. Xu, M.; Xing, Y.; Gui, X.; Cao, Y. Flotation Kinetics of Easy-to-Float Fine Coal. *Energy Sources Part A Recovery Util. Environ. Eff.* **2017**, *39*, 1978–1983. [\[CrossRef\]](#)
13. Safari, M.; Hoseinian, F.S.; Deglon, D.; Leal Filho, L.S.; Souza Pinto, T.C. Investigation of the Reverse Flotation of Iron Ore in Three Different Flotation Cells: Mechanical, Oscillating Grid and Pneumatic. *Miner. Eng.* **2020**, *150*, 106283. [\[CrossRef\]](#)
14. Carelse, C.; Manuel, M.; Chetty, D.; Taguta, J.; Safari, M.; Youlton, K. The Flotation Behaviour of Liberated Platinum Group Minerals in Platreef Ore under Reduced Reagent Conditions. *Miner. Eng.* **2022**, *190*, 107913. [\[CrossRef\]](#)
15. Safari, M.; Hoseinian, F.S.; Deglon, D.; Leal Filho, L.; Souza Pinto, T.C. Impact of Flotation Operational Parameters on the Optimization of Fine and Coarse Itabirite Iron Ore Beneficiation. *Powder Technol.* **2022**, *408*, 117772. [\[CrossRef\]](#)
16. Vinnett, L.; Waters, K.E. Representation of Kinetics Models in Batch Flotation as Distributed First-Order Reactions. *Minerals* **2020**, *10*, 913. [\[CrossRef\]](#)
17. Schuhmann, R., Jr. Flotation Kinetics. I. Methods for Steady-State Study of Flotation Problems. *J. Phys. Chem.* **1942**, *46*, 891–902. [\[CrossRef\]](#)
18. Kabemba, A.M.; Mutombo, K.; Waters, K.E. A Predictive Geometallurgical Framework for Flotation Kinetics in Complexes Platinum Group Metal Orebodies: Mode of Occurrence-Based Modification of the Kelsall Model Using Particle Swarm Optimization. *Minerals* **2025**, *15*, 701. [\[CrossRef\]](#)
19. Murhula, E.; Hashan, M.; Otsuki, A. Effect of Solid Concentration and Particle Size on the Flotation Kinetics and Entrainment of Quartz and Hematite. *Metals* **2023**, *13*, 53. [\[CrossRef\]](#)
20. Bahrami, A.; Mirmohammadi, M.; Ghorbani, Y.; Kazemi, F.; Abdollahi, M.; Danesh, A. Process Mineralogy as a Key Factor Affecting the Flotation Kinetics of Copper Sulfide Minerals. *Int. J. Miner. Metall. Mater.* **2019**, *26*, 430–439. [\[CrossRef\]](#)
21. Safari, M.; Deglon, D. Evaluation of an Attachment–Detachment Kinetic Model for Flotation. *Minerals* **2020**, *10*, 978. [\[CrossRef\]](#)
22. Hoseinian, F.S.; Safari, M.; Deglon, D. Ion Flotation Kinetic Predictions Using Empirical and Phenomenological Models. *Miner. Eng.* **2024**, *210*, 108645. [\[CrossRef\]](#)
23. Yianatos, J.; Vallejos, P. Challenges in Flotation Scale-up: The Impact of Flotation Kinetics and Froth Transport. *Miner. Eng.* **2024**, *207*, 108541. [\[CrossRef\]](#)
24. Ek, C. Flotation Kinetics. In *Innovations in Flotation Technology*; Springer: Dordrecht, The Netherlands, 1992; pp. 183–210.
25. Zhang, H.; Liu, J.; Cao, Y.; Wang, Y. Effects of Particle Size on Lignite Reverse Flotation Kinetics in the Presence of Sodium Chloride. *Powder Technol.* **2013**, *246*, 658–663. [\[CrossRef\]](#)
26. Bu, X.; Xie, G.; Peng, Y.; Ge, L.; Ni, C. Kinetics of Flotation. Order of Process, Rate Constant Distribution and Ultimate Recovery. *Physicochem. Probl. Mineral. Process.* **2017**, *53*, 342–365.
27. Neethling, S.J.; Brito-Parada, P.R.; Hadler, K.; Cilliers, J.J. The Transition from First to Zero Order Flotation Kinetics and Its Implications for the Efficiency of Large Flotation Cells. *Miner. Eng.* **2019**, *132*, 149–161. [\[CrossRef\]](#)
28. Xu, W.; Liu, Q.; Dai, S.; Liu, J.; Li, S.; Xing, Y.; Gui, X. Machine Learning-Based Prediction of Multivariate Condition Flotation Kinetics. *Powder Technol.* **2026**, *470*, 122007. [\[CrossRef\]](#)
29. Niemi, A.J. Role of Kinetics in Modelling and Control of Flotation Plants. *Powder Technol.* **1995**, *82*, 69–77. [\[CrossRef\]](#)
30. Yalcin, E.; Kelebek, S. Flotation Kinetics of a Pyritic Gold Ore. *Int. J. Miner. Process.* **2011**, *98*, 48–54. [\[CrossRef\]](#)
31. Vizcarra, T.G.; Harmer, S.L.; Wightman, E.M.; Johnson, N.W.; Manlapig, E. V The Influence of Particle Shape Properties and Associated Surface Chemistry on the Flotation Kinetics of Chalcopyrite. *Miner. Eng.* **2011**, *24*, 807–816. [\[CrossRef\]](#)
32. Yuan, X.-M.; Palsson, B.I.; Forssberg, K.S.E. Statistical Interpretation of Flotation Kinetics for a Complex Sulphide Ore. *Miner. Eng.* **1996**, *9*, 429–442. [\[CrossRef\]](#)
33. Zhu, S.; Xia, M.; Chu, Y.; Khan, M.A.; Lei, W.; Wang, F.; Muhmood, T.; Wang, A. Adsorption and Desorption of Pb(II) on L-Lysine Modified Montmorillonite and the Simulation of Interlayer Structure. *Appl. Clay Sci.* **2019**, *169*, 40–47. [\[CrossRef\]](#)
34. Metso Metso HSC Chemistry. Available online: <https://www.metso.com/portfolio/hsc-chemistry/> (accessed on 18 March 2026).
35. Ross, V. Key Aspects of Bench Flotation as a Geometallurgical Characterization Tool. *J. South. Afr. Inst. Min. Metall.* **2019**, *119*, 361–367. [\[CrossRef\]](#)
36. Amini, E.; Bradshaw, D.J.; Xie, W. Influence of Flotation Cell Hydrodynamics on the Flotation Kinetics and Scale up, Part 1: Hydrodynamic Parameter Measurements and Ore Property Determination. *Miner. Eng.* **2016**, *99*, 40–51. [\[CrossRef\]](#)

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