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## Effect of pyrite content of feed and configuration of locked particles on rougher flotation of copper in low and high pyritic ore types

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### ABSTRACT

Misreported pyrite into copper concentrates dramatically declines copper grade and recovery. Copper flotation can be also more complicated if flotation feed comes from an elevated-pyritic copper ore. In this investigation, the effect of two different ore types (high pyritic and low pyritic feeds) was studied on rougher stage of industrial copper flotation circuit. Samples were taken from different streams and the structure of chalcopyrite within the pyrite and non-sulfide gangue minerals was examined in various size fractions for mentioned ore types. Results indicated that 72% and 56% of the total floated pyrite was transferred to concentrate in first four cells in the low and high pyritic feeds, respectively. Whereas, this proportion for floated SiO<sub>2</sub> in last ten cells was detected as 72% and 71%, respectively. A detailed interpretation of the effect of locked particles in different size fractions on rougher flotation cells is studied from industrial point of view.

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### 1. Introduction

Flotation is a significant mineral processing technique used for selective separation of valuables from gangue minerals. Industrial practice of froth flotation is widely recognized to be a complex process [1]. In addition to hydrodynamics and mechanical parameters of flotation cells, surface properties and crystal structure of particles are the most crucial factors which affect the flotation process [2]. The efficiency of a flotation circuit operation depends on several factors related to minerals' nature and structure (mineralogy, morphology and particle size) as well as type of instrumentation and operational variables (air flowrate, pulp density and reagent type and dosage) which have been extensively investigated both theoretically and practically [3–15]. Also, each process has its own specifications and properties in operation according to its particular constraints and restrictions which should be separately and precisely investigated in order to achieve the maximum possible recovery and efficiency in the plant.

Aside from hydrodynamics circumstances and physicochemical properties, particle size is an important parameter in flotation pro-

cesses. The relationship between particle size and flotation recovery has been the subject of several studies. It is reported that the behavior of fine and coarse particles in flotation follows different trends [16,17]. In the case of copper flotation, maximum recovery can be achieved at the intermediate particle size range of 50–70 μm [18–20]. Obviously, particles of extreme sizes (too coarse and too fine) are more difficult to be recovered by froth flotation compared to the intermediate particles. Mineral liberation also plays an important role in efficiency of the flotation processes as demonstrated by studies on particles-bubble detachment as well as flotation kinetics rate of liberated and locked particles [21]. As the particle size decreases, floatability of ore diminishes gradually due to the increase in the number of particles per unit weight (bubble surface coverage restriction), deteriorating conditions for particle-bubble collision (such as dwindled inertial force), and the changes in surface properties such as surface oxidation [17,22,23]. Sudden decrease in flotation recovery above the optimum particle size was suggested to be due to either lack of liberation of valuable minerals or reduction in ability of bubbles to lift up the heavy coarse particles [17,24–26]. According to previous reports, most of the copper loss which is misreported to flotation tailings virtually in all operating concentration plants occurs in particle sizes below 20 μm (50%) and over 150 μm (30–40%) [27–29].

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In practice, non-valuable iron sulfides (mostly pyrite) are often associated with valuable sulfide minerals (chalcopyrite, covellite and chalcocite) in copper flotation concentrate. These iron sulfides play a significance role especially in porphyry copper flotation concerning their devastating effect on quality of product (concentrate grade) and efficiency of the process (mineral recovery) [30,31]. Thus, the efficient separation of pyrite from copper sulfide minerals during the flotation process is vital [32]. In copper flotation plants, the general strategy is to maximize the recovery of copper minerals along with floating some pyrite at the rougher stage (non-selective bulk flotation) followed by further rejection of iron sulfides at the cleaner stage after regrinding. Despite an increase in number of liberated particles after regrinding, rejection of pyrite can be difficult if fresh pyrite surfaces become activated by dissolved copper ions [33]. This process would be even more complicated if the flotation feed contains higher grades of pyrite, which may result in a plunge in copper grade and overall recovery. Consequently, most flotation circuits processing complex sulfide ores, use regrinding (to liberate pyrite and chalcopyrite) and depression strategies to minimize the pyrite flotation by means of aeration, adding sodium cyanide or sulfite and applying high pH values ( $\text{pH} > 10$ ). Leppinen pointed out that adsorption of xanthate onto the pyrite surface without activation reaches a maximum at pH of 5 and a minimum at pH of 7 [34]. Flotation results presented that around 80–90% of pyrite can be recovered through flotation (without activation) at pH 4–5, while recovery is extremely lower at other pH values [35].

Basically, in flotation of complex sulfide ores, a higher pH is generally used to separate valuable sulfide minerals from pyrite or pyrrhotite in the presence of xanthate collectors [36]. Several works extensively discussed the impact of pyrite on copper flotation [37–42]. It was reported that beside its hydrophobic nature in fresh surfaces, the surface reactions and electrochemical processes promote pyrite activation, which was indicated as the main reason for its floatability [43]. It was also shown that both activated and bare pyrite surfaces interact with thiol collector molecules such as xanthates [34]. Trahar et al. indicated that under alkaline conditions, recovery of pyrite is mostly dominated by entrainment [44]. Nevertheless, interactions between sulfide minerals may also promote pyrite floatability. Kawatra and Eisele suggested that flotation of liberated pyrite in coal cleaning flotation plants is not an important recovery mechanism compared to that of entrainment and mechanical contiguity with floatable particles [45]. Bulut and Goktepe investigated the effect of copper activation on five different high grade pyrite samples [46]. They concluded that although the flotation of bare pyrite was achieved at acidic pH ranges, they also can be floated at basic pH regions in the presence of copper ions. Mielczarski et al. studied the adsorption of xanthate on chalcopyrite [47]. They showed that xanthate molecules should be in contact with a fresh chalcopyrite surface in order to produce cuprous complex and dixanthogen as a hydrophobic site. If a mineral surface is covered with hydrophilic species (i.e., iron hydroxides) a larger amount of collector will be required to remove the hydrophilic surface species and produce hydrophobic surface on particles. It was also represented that in copper complex ores with higher locked copper particles, separation is more difficult even in cleaner flotation stage after regrinding [48]. Therefore, it is essential to take the role of pyrite into account in the copper flotation process, not only for being aware of its extremely detrimental effects on copper grade and recovery but also particularly from environmental point of view, to reduce  $\text{SO}_2$  emission through further smelting processes.

Despite decades of research on activated and bare pyrite surface in porphyry copper flotation process, there is still a considerable lack of industrial investigation and data in this area. The motivation of the current study is to understand floatation response of

high and low pyritic copper ore types as well as a detailed industrial study in variety of size fractions regarding mineralogical aspects and liberation factors. In addition, a comparison of results in terms of copper grade and recovery for the mentioned ore types have been provided. In this work, mineralogical and metallurgical data are analyzed and reconciled in various size classes for two both feeds of low and high pyritic contents. In addition, locked and liberated particles of concentrate streams of each cell in rougher circuit have been evaluated with the aim of transferring liberated chalcopyrite concentrate directly to recleaner section instead of cleaner cells to avoid its unnecessary regrinding. To determine the role of free and locked minerals (chalcopyrite, pyrite and silicate) in two mentioned feed types, liberation degree analysis was established. Moreover, contribution of each mineral in different particle size fractions of concentrate streams for each cell has been investigated in the rougher flotation stage.

## 2. Material and methods

### 2.1. Sarcheshmeh copper concentrator

Sarcheshmeh concentration plant is the largest copper concentrator in Iran which is located in Kerman province. Reserve contains one billion tons of 0.70% copper and 0.03% molybdenum in average. Its flotation circuit is producing 50,000 tons copper concentrate per month with an average grade of 28–32% of copper in final recleaner banks. A schematic diagram of flotation circuit of Sarcheshmeh concentrator is presented in Fig. 1. Rougher flotation cells comprise eight rows of flotation machines, and consist of two sections. Each section has four rows of flotation cells. The volume of each flotation cell is  $8.5 \text{ m}^3$ . Subsequently, each row includes fourteen mechanical flotation cells which are stepped in three units (4 cells–5 cells–5 cells) gravitationally [49]. The combined rougher flotation circuits are designed to process ground ore feed at a rate of 43,360 dry metric tons per day. The concentrate of the first stage of flotation is reground and its tailing constitutes the main portion of the plant's final tailings. A concentrate with an average grade of 30% Cu is obtained after cleaning and recleaning stages. Total recovery of the plant varies between 88% and 90% depending on the operating conditions and the feed ore type [50,51].

Rougher cells in this plant are expected to produce a concentrate of 7.6% Cu with a recovery of 96%. In fact, rougher tailings contain 80% of final tailings of the flotation circuit. Therefore, it is very important to reach a maximum possible recovery in rougher banks in order to avoid misreporting of valuable copper

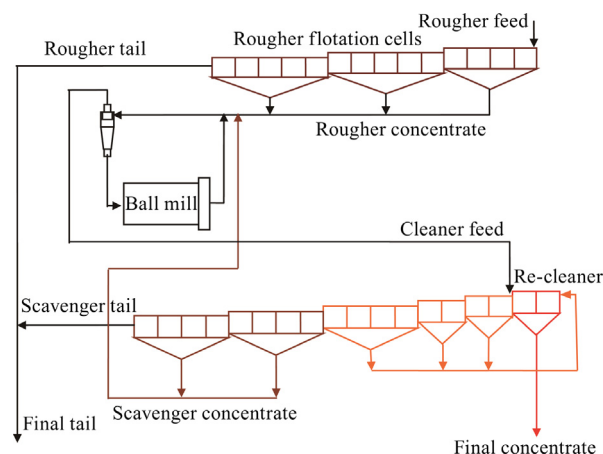


Fig. 1. Flotation circuit of Sarcheshmeh copper concentrator.

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