RFI 090
Pebble Project EIS

Request for Information

Title/Subject:	**In Pit Crushing and Conveying**
Requestor:	**AECOM**
Date Transmitted:	**10/5/2018**
Recipient:	**Pebble Limited Partnership (PLP)**
Response Requested by:	**10/12/2018**
Rationale:	In pit crushing and conveying is evaluated in PLP's May 25, 2018 Response to RFI-032 Project Options. A rough concept of the option is presented with an argument that the option offers no environmental advantages and several disadvantages. USACE presented a discussion of the option in its September 2018 draft of Appendix B Alternatives Development and concluded it should be dismissed from detailed evaluation in the EIS. Cooperating agency comments on the draft of Appendix B request additional analysis of the option: *It is not clear why in-pit crushing and conveying would create the need for a separate PAG waste storage facility. In pit crushing and conveying is common technique. We recommend providing more explanation that includes the estimated amount of PAG waste that would be generated, why the PAG waste cannot be included in the proposed pyritic TSF/PAG waste facility, and the estimated increased footprint of the proposed TSF/PAG facility or a new facility under this option.* *In-pit crushing should be included as an option in the alternatives analysis and evaluated in the EIS, since in-pit crushing and conveying are probably less environmentally damaging than other alternatives involving transporting, crushing, and handling ore. This option should be evaluated and compared to Option MNG-004, which involves haul trucks transporting ore from the pit to a crusher located away from the pit.*
Describe the Information Requested and Level of Detail:	Future versions of Appendix B will include Figure 1 from PLP's May 25 response to RFI-032. Please provide the following information to allow additional evaluation of this option: • Quantification of the amount of additional PAG waste, • Analysis of space available in the proposed pyritic TSF and expansion potential, • Quantification of the increased footprint and wetlands impacts from expansion of the pyritic TSF or a new PAG storage facility, and • Estimation of the change in emissions for trucking during mining and site reclamation and closure and operation of the IPCC.

Recipient Response Form

Date Received from USACE:	Click here to enter text.
Response from Recipient (Describe Information Requested to the Level of Detail Requested; Provide Attachments as Needed):	Click here to enter text.
List Number and Type of Response	**RFI_090_In_Pit_Crushing_and_Conveying.pdf** **paricheh et al 2018.pdf**

Attachments:	
Date Returned to USACE:	Click here to enter text.

AECOM Intake Form

Date Response was Received:	**10/29/2018**
Received by:	**AECOM**
Describe any Follow-up Related to this RFI (Communications, Clarifications):	Click here to enter text.

From: James Fueg, Pebble Limited Partnership

To: Shane McCoy, US Army Corps of Engineers

Date: October 28th, 2018

The questions presented in RFI 090 on in pit crushing and conveying are addressed below:

Cooperating agency comments on the draft of Appendix B request additional analysis of the option:

> *It is not clear why in-pit crushing and conveying would create the need for a separate PAG waste storage facility. In pit crushing and conveying is common technique. We recommend providing more explanation that includes the estimated amount of PAG waste that would be generated, why the PAG waste cannot be included in the proposed pyritic TSF/PAG waste facility, and the estimated increased footprint of the proposed TSF/PAG facility or a new facility under this option.*
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Overview

To respond to RFI 090, PLP used the proposed mine plan to develop a preliminary open pit layout that would incorporate in pit crushing and conveying (IPCC). The proposed mine plan and plan adjusted for IPCC are shown in Figures 1 and 2. The IPCC plan incorporates a patio on the western end of the pit at the 700 ft elevation, or approximately 400 ft from surface. The crushers would be located here in approximately Year 14 of the 20 year mine life.

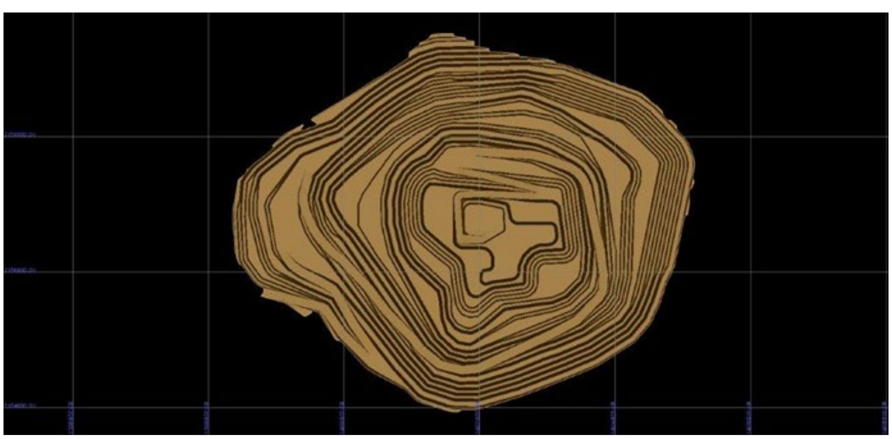

Figure 1: Proposed mine plan

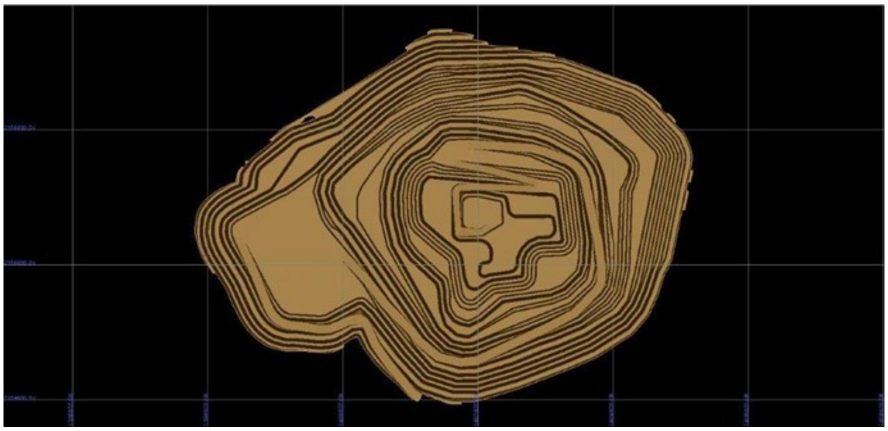

Figure 2: Mine plan adapted for IPCC

Table 1 compares the tonnages of the two mine plans.

	Mill feed	Waste	Total	Waste backfill
	Million tons	**Million tons**	**Million tons**	**Million tons**
Pit without IPCC	1,291	153	1,443	129
Pit with IPCC	1,301	223	1,524	200
Difference	10	71	81	71

Table 1: Pit comparison

Table 1 demonstrates the significant increase in waste rock required to expand the pit to accommodate the IPCC, with the strip ratio increasing from 0.12:1 to 0.17:1. This preliminary assessment assumes all of this additional waste rock must be placed in the pyritic tailings storage facility (TSF), increasing the capacity by approximately 25%. Further, at closure, this additional waste must be returned to the open pit.

The IPCC pit would increase the pit footprint by 140 acres, of which 45 acres would be wetlands. To store the additional PAG waste, the pyritic TSF would need two additional 25-foot raises. This would push the three embankments out by an estimated 125-150 feet, resulting in an increase in the footprint of approximately 91 acres, of which 26 acres are wetlands.

PLP then compared the haulage truck hours between the two scenarios. Hauling to the in pit crushers could only commence after the patio was excavated, the site prepared, and the crushers relocated. For this analysis, it was assumed that haulage would commence after two additional benches had been mined. Table 2 compares the total truck hours for the two scenarios.

Estimated Truck hours		First 14 years	Last 6 years	Total
Mill feed	Without IPCC	702,169	597,489	1,299,659
	With IPCC	706,750	425,669	1,132,419
	Difference	***4,580***	***(171,820)***	***(167,240)***
Waste	Without IPCC	87,828	29,814	117,642
	With IPCC	140,684	29,814	170,499
	Difference	***52,856***	***-***	***52,856***
Waste backhaul	Without IPCC	-	-	171,350
	With IPCC	-	-	265,086
	Difference	***-***	***-***	***93,735***
Net change in truck hours				(20,648)

Table 2: Truck hour comparison

As shown in Table 2, the IPCC alternative would only reduce the haulage truck requirements by approximately 21,000 hours, or less than 1.5%. In addition to the small reduction in truck hours, the conveying system would draw an incremental 15 MW of power. Assuming 75% utilization of the IPCC system, this system would require almost 600,000 MWh of energy over its 6 years of operation. The haulage trucks are equipped with 2.5 MW engines and thus even assuming full power draw during the full haulage cycle, the IPCC alternative consumes significantly more energy than the proposed alternative.

In conclusion, an IPCC alternative would:

- Significantly increase the waste tonnage to be handled, likely requiring either an expanded footprint of the existing pyritic TSF or construction of a stand alone waste rock facility;
- Increase the footprint of the open pit and pyritic TSF by 231 acres, including 71 acres of wetlands; and
- Increase the mine haulage energy consumption over the project life.

Even ignoring the financial implications, this conclusion corroborates the initial analysis completed by PLP. For further corroboration, PLP conducted a search of published literature. This search identified a paper (Paricheh et al, 2016), in which the authors concluded IPCC is best installed at a depth of 490 m (1,600 ft) and after 17 years of mine life.

Questions

- *Quantification of the amount of additional PAG waste,*

  Implementation of IPCC results in a pit containing an additional 81 million tons, of which 71 million tons are PAG waste.

- *Analysis of space available in the proposed pyritic TSF and expansion potential,*

  Based on this preliminary analysis PLP believes the additional waste could be stored in the proposed pyritic tailings facility, but the size would have to be increased to accommodate two additional raises to the embankments.

- *Quantification of the increased footprint and wetlands impacts from expansion of the pyritic TSF or a new PAG storage facility, and*

  The increase in the pit and pyritic TSF footprints would total approximately 231 acres, of which 71 acres are wetlands.

- *Estimation of the change in emissions for trucking during mining and site reclamation and closure and operation of the IPCC.*

- IPCC would result in a net reduction of approximately 21,000 hours, or approximately 52,500 MWh of energy consumed conservatively assuming the trucks operate at the full power draw. The IPCC conveyor system would be expected to draw an 600,000 MWh over six years of operations (in addition to the energy consumed by the trucks hauling ore to the crushers and waste to the pyritic TSF), resulting in a substantial net increase in energy consumption.

Reference:

Paricheh, M, Osanloo, M, & Rahmanpour, M, 2016. “A heuristic approach for in-pit crusher and conveyor system’s time and location problem in large open-pit mining”, International Journal of Mining, Reclamation and Environment, October 2016, p 1-21.

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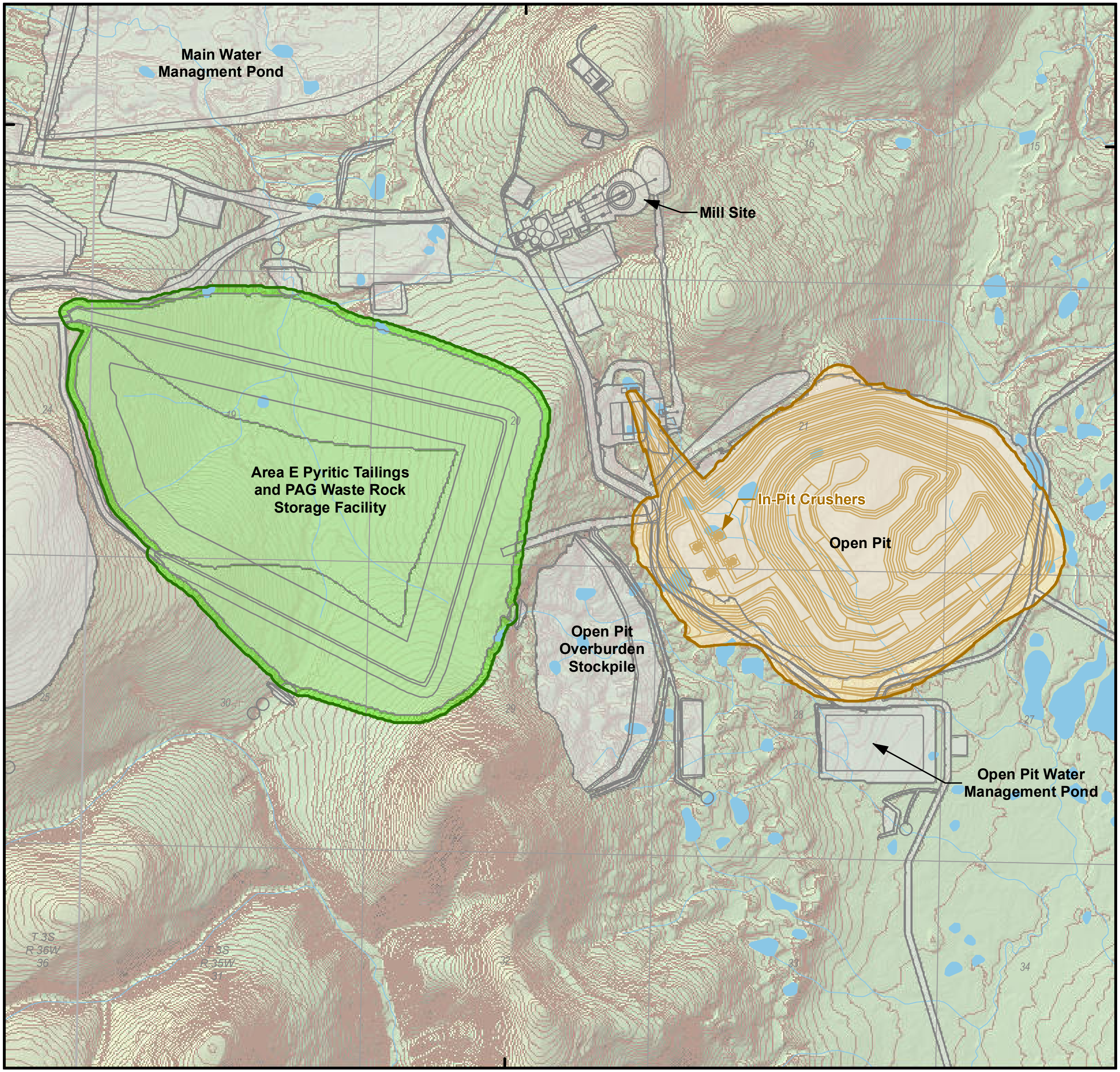

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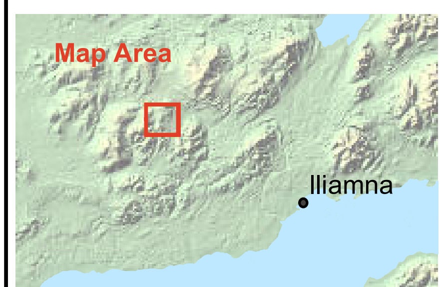
FIGURE 1

IPCC Pit Outline

- Current Mine Layout Design
- Pit concept with In-Pit Crushing and Conveying
- Expanded Waste Rock Storage Facility for In-Pit Crushing and Conveying



Miles: 0 — 0.45 — 0.9

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Alaska State Plane Zone 5 (units feet)
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A heuristic approach for in-pit crusher and conveyor system's time and location problem in large open-pit mining

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ABSTRACT

Cost efficiency and high reliability of In-Pit Crushing and Conveying (IPCC) system make it more appealing to be used in deeper open-pit mining activities. Determination of the optimum time and location (OT–OL) for applying the IPCC system were always a challenge. Application of mathematical programming approaches suffers from reduced computational efficiency due to a lot of decision variables. The situation will be worse; by adding the IPCC's time and location variables to these models. Also, the problem involves a lot of variables that each one of them is a function of others. To tackle these, this paper provides a new heuristic approach to find a good quality solution hierarchically (here in two levels). The problem was broken into two parts (i.e. OT and OL). The model works in a discrete time horizon. It has been verified by the data gathered from Sungun copper mine of Iran. Our computational results show that the system is efficient for developed countries with a stable and low interest rate.

ARTICLE HISTORY

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KEYWORDS

Optimum location; optimum time; open-pit mining; in-pit crusher; IPCC system

1. Introduction

Nowadays, as a result of economy of scale and geological conditions of mineral deposits, cut-off and the average grade of deposits are decreased. Compared with the last century, the stripping ratio in open-pit mines has been increased. On the other hand, the world population in the year 2050 will increase to 10 billion which increases the demand for raw material [1]. Surface mines deliver the majority of minerals, and it accounts for about 90% of the total mineral production. Easily accessible deposits have been depleted; and now is the time to reach deposits laid at depth or in more remote and difficult regions [2]. Large-scale mining at great depths is only possible through open-pit mining method. In the 1950s, the depth of surface mines was around 300 m. Now, Chuquicamata in Chile is working at a depth of approximately 1100 m [3].

Today, more than 80% of open-pit mines in the world use the shovel-truck system for loading–hauling operation. The conventional shovel-truck system is popular due to its flexibility. The system can start with a small fleet of trucks, and trucks can be added to the fleet as the production ramps up. Also, the new open-pit mining operations often start with the use of trucks because the crushing plant is relatively close to the mining face at this stage [4]. This situation changes once the pit becomes deeper. With increasing depth, haulage distance increases and the number of loads per truck decrease.

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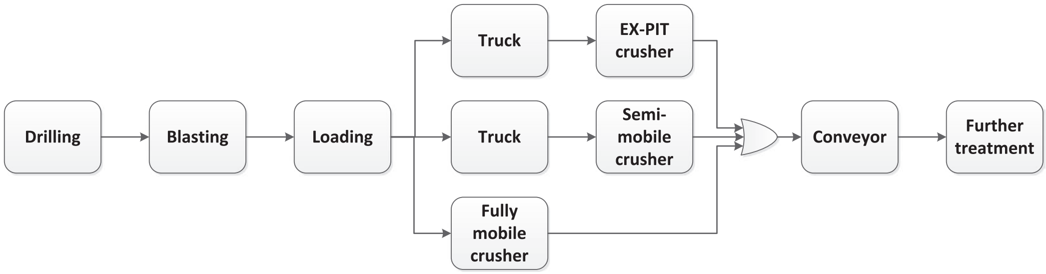

Figure 1. Typical transportation systems in open-pit mines.

Therefore, the number of trucks should be greater than before. Trucks need appropriate technical support, not only in term of stores, shops, tanks and parking, but also additional machinery such as water wagons, dozers, front-end loaders, towers and cranes [2]. As a result, road maintenance, fuel, tyre and depreciation costs per ton increase [5–7]. Because of these, an economical travel distance for trucks is only a few kilometres (about 3.2 km) [8]. These conditions have resulted in open-pit mining at great depth (300–1000 m) with traditional transportation systems to face some technical and economic problems.

Optimization of truck usage by applying a truck dispatching system and using the larger trucks are two methods for reducing haulage costs in open-pit mines [9,10]. As a result, over the past five decades, mining equipment has steadily increased in size and complexity. However, there is a limited understanding of how equipment size affects various aspects of mining operations. This means that the cost/benefit of using the next generation of larger equipment is not clear [11]. The other way of reducing the haulage costs is to shorten the truck haulage distance by bringing the truck dump point into the pit. It needs 'In Pit Crushing-Conveying (IPCC) and trucks' system. Cost efficiency and high reliability of IPCC systems compared to conventional truck-shovel systems make it more appealing to be utilized in deep open-pit mining activities. Figure 1 shows different haulage systems in open-pit mines. In-pit crushers are classified as semi-mobile and fully mobile operation. Given the features of open-pit mines, semi-mobile systems are best suited for these operations. This paper studies the application of the IPCCs as a solution for future open-pit mining.

Most of the world's open-pit mines used pure truck systems for transportation at the beginning and after a few years of mining operation when trucks were not affordable, some of them such as Chuquicamata and Bingham Canyon decided to use IPCC systems instead. Chuquicamata used the IPCC system in the 1980s and 1990s for transporting ore and waste, respectively [7]. Bingham Canyon's conventional ore transport system was changed to IPCC in 1988 [12]. The decision is due to the fact that at the beginning, the pure truck system is more economical. Determination of the optimum time and optimum location (OT–OL) for application of the IPCC system is a challenge that is less studied. The optimum depth is where the transfers of a shovel-truck system into the IPCC system with more efficiency and less cost can be done. Similarly, the optimum time is the year that shovel-truck system is changed into an IPCC system.

Optimization during the mine planning and design phase of open-pit mining projects uses the methodologies based on operation research (OR) concepts. Some of these concepts involved: determination of ultimate pit limit [13,14], short-term and long-term block sequencing [15,16], uncertainty and risk analysis [17] and determination of equipment specification (number and size) in the short-term planning [10,18–20]. The objective of this paper is to develop a generic optimization methodology based on OR concepts to identify the suitable time and location of IPCC system in open-pit mines. At the end, the model was verified by the data gathered from Sungun copper mine (SCM) of Iran.

2. Literature review

IPCC system has been known in the mining industry for many decades. The idea of in-pit crushing was introduced in 1956 in Germany. More than 200 mobile and semi-mobile crushing systems have been installed worldwide [4]. A review on the advantages/disadvantages and operating techniques of crushers, feeders, conveyors, and stacking systems was reported by Hreber and Jeric (1997) [21]. The paper studies those operations that used large-scale continuous haulage systems. Barua and Lanergan (1985) developed a computer programme that compares various conveyor layouts with regard to the system output. They believe that to maximize the economic benefit of in-pit conveying, one must minimize the cost of the flatter slope expressed in terms of additional stripping or tied up ore [22]. Dos Santos and Stanisic (1987) evaluated the application of high-angle conveyor in Majdanpek copper mine with IPCC system [23]. Kammerer (1988) reviewed the special design features and status of IPCC system in Bingham Canyon [12].

There are some researches on the topic of locating the IPCC in open-pit mines. Sturgul (1987) developed a simulation-based programme to determine the optimum location of in-pit crusher system [24]. Koehler (2003) showed that continuous mining systems such as in-pit crushing systems are most cost-effective in the case of large capacity, long mine life, deep pits and long haulage distance [4]. Yu Changzhi (2003) studied the optimum level where the shovel-truck haulage system is transferred into IPCC systems. He considered relocation of an in-pit crushing system and the impact of the transferring point when the costs change. This paper focuses on the problem transiently and also some affecting parameters on the optimization are not considered [25]. Zimmermann (2006) believes in IPCCs as a chance for better and cheaper production. In this context, considerations regarding applications, case studies and the economic effects of fully mobile crushing and conveying systems were assessed [26]. Konak et al. (2007) discussed the effects of pit geometry and mine access requirements on optimum crusher location that is mainly based on the establishment of minimum haulage distance. They established a trial and error process and applied their method in an aggregate mine [27]. Turnbull and Cooper (2009) and Morrison and Lourel (2009) evaluated the IPCC system and they tried to document the options that might be employed for dig side, transport and dump side operations. They rank them in terms of system applicability. They believed that when average truck cycle time is more than 25 min; IPCCs are likely to be economic. Also, it is noted that IPCC is ideally suited for new operations or to the expansion of existing operations, rather than an existing steady-state operation. They also studied types of IPCCs that are best suited for a particular mining application [28,29]. Aghajani et al. (2009 and 2011) believe that the selection of most appropriate material handling system must be based on economic, social, environmental and technical issues [19,20]. Rahmanpour et al. (2014) studied the effective factors on the determination of a suitable location of an IPCC and investigated its locating as a single hub location problem. The main concern is to minimize haulage costs regarding the environmental concerns [30].

3. Problem definition

Nowadays, decreasing the cost is a must in deep mining operations. If the IPCC system fails to improve the economic condition it would not be used. So, choosing the OT–OL of IPCC definitely reduces the total hauling cost (i.e. investment and operating). A sub-optimal selection will lead to higher production costs. Therefore, the problem is an economic optimization problem concerning the transportation cost [31]. Selection of OT–OL for transferring from pure truck to the IPCC system depends on various parameters and mainly on economic, technical and environmental issues.

Discounted cumulative cost curve tendency of both pure truck and IPCC systems are shown in Figure 2. As shown in the figure, the initial investment of IPCC system is more than the pure truck systems. The IPCC system's equipment is purchased at the beginning while the truck system can start with a few numbers of trucks, and trucks can be bought gradually. Also, cumulative cost curve of IPCC has a lower slope compared to the pure truck system. The difference between the two curves

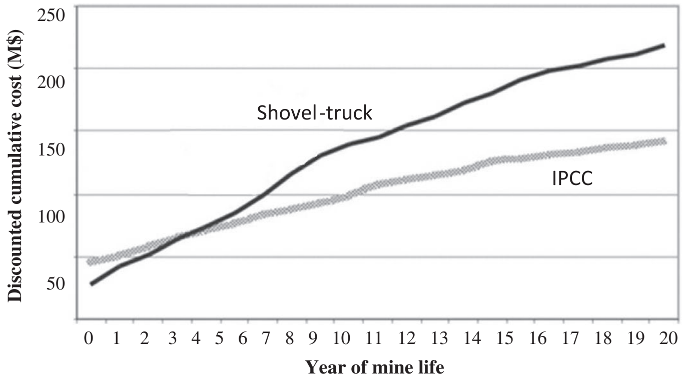

Figure 2. Discounted cumulative cost of truck-shovel system vs. IPCC system [25].

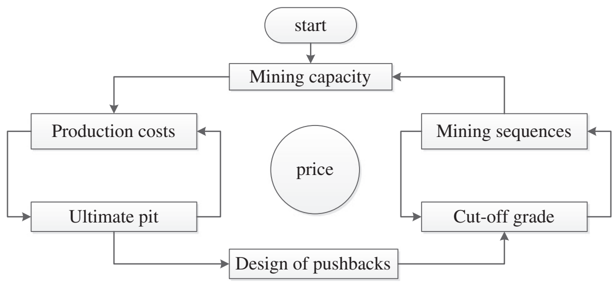

Figure 3. Open-pit long-term production planning variables [35].

is due to the difference between incremental rates of haulage cost curves of the two systems. Over the time, two curves move away from each other. It means, for longer life mining operations, IPCC systems will be more economical, especially in the case of long haulage distances and deep pits [32].

According to the open-pit mine planning process, pit outline with the highest value cannot be determined until block values are known. The block values are not known until the mining sequence is determined; the mining sequence cannot be determined unless the pit outline is available [33]. This is a large-scale mathematical optimization problem which could not be solved currently using commercial packages. The most common solution approach to the problem is dividing it into sub-problems. For optimization purposes, these sub-problems which include ultimate pit limit, yearly mine schedules, and cut-off grade strategies are interacting in a circular fashion (Figure 3). The optimization is started with some assumptions of initial production capacities and estimates for related costs. The commodity price is the heart of the process. Then, using the economic block values the ultimate pit limit is determined based on Graph theory or maximum flow algorithm [13,34]. Within the ultimate pit, pushbacks (smaller pits) are designed as guides during the yearly mine production plan. Prior to mining schedule, the cut-off grade strategy must be determined to discriminate between ore and waste, and further, to determine how the individual blacks should be processed. The problems for a given mine can be formulated using large-scale LP/IP models containing over 100,000 variables and 50 to 100,000 constraints individually [35].

Application of mathematical programming approaches for solving the production planning problem suffers from reduced computational efficiency due to many decision variables. The situation will be worse by adding the IPCC's time and location variables to these models. The block values change using the IPCC system in the middle of mine life. Following that, the pit outline and mining sequences change. While pit outline and mining sequences are the main affecting factors in the problem (i.e. OT–OL), as seen optimization of the transferring time and depth from shovel-truck operation to the IPCC system deals with many interdependent variables.

In the case of the open-pit mining operation, relocation of the in-pit crushing station is required in the periods when the haulage distances become uneconomic for trucks. Crusher location indicates the haulage length for both parts (conveyor and truck) of IPCC system and also the capacity of the haulage system. It means that IPCC's capital and operating cost depend on the in-pit crusher locations strongly [36]. Therefore, selecting the optimum location and the optimum time for relocation of in-pit crusher is an integral part of the optimizing an IPCC system. On the other hand, deciding about when to start the system is an economic optimization problem and it needs an accurate estimation of aforementioned incremental rates of haulage cost curves of the two systems (IPCC and pure truck) and periodic investment over the mine life. It means the optimum time of IPCC depends on the optimum locations of the crusher through the mine life. So, the outputs of the locating problem (i.e. locations and time of relocations) are the required parameter of the timing problem, but not directly. For these reasons, integrating the two problems (timing and locating problems) into one more general model is not possible. The decision should be made hierarchically (here in two levels).

4. Optimal time and location models development (solution technique)

To overcome the problems presented in the previous section, the problem was broken into *two main parts (OL and OT)*.Then it is solved using heuristic trial and error technique. In the next, more details are at hand to clear how both models share information among themselves.

The first part (OL) is the design of the IPCC, which involves determination of the conveyor exit method and also in-pit crusher locations through the mine life. Then, one primary concern with the installation of an IPCC system is how the conveyors exit the pit. There are three alternatives by which the conveyors can exit the pit: (1) tunnel, (2) dedicated (generally steep) conveyor ramp and (3) via an existing haul road [28,29]. After this stage, considering the mine production plan, the OL model decides a different location for the different periods of the mine life. The inputs (parameters) and outputs (decision variables) of the OL model will define in the next sub-section.

The second part (OT) determines the best time for applying the IPCC system which cannot be performed unless the in-pit crusher locations in all of mine life are known. When the locations and time for relocations are identified, the initial and periodic investment and also operating costs of the IPCC system will be estimated precisely. The location of this type of system reduces the truck transportation costs but incurs in a location cost. The OL model helps us estimate these costs and the time of investment.

Firstly, the OL model finds out the crusher locations for remaining years, taking into accounts the time of applying the system. Then, the OT model evaluates different scenarios (time) for applying the IPCC system and chooses the best option with regard to the time value of money. The initial and periodic location costs are paid in different applying time. Indeed, the crusher locations are implicitly the inputs of OT model. The time which the OT model decides to start the system is the main output. Afterward, the decision-maker should have a drawback to the OL model results and find the best locations with respect to the applying time. These steps are only the first iteration of the optimization procedure.

Now, assuming the locations and time calculated in the first iteration, a new extraction cost of blocks, ultimate pit limit and extraction sequences should be identified. Then, using the updated database the second iteration of the procedure is run similarly. These steps are repeated until further improvement of the NPV of the project is not possible.

It is assumed that for a given open-pit mine, the following technical parameters have already been determined:

- mineable reserve and topography of the area,
- life of mine, the final pit limit and the geometry of the pit which are usually determined by the characteristics and the capacity of the equipment utilized,

- production per year and the annual schedule of production,
- the location of the ex-pit crusher and the external dumping site for waste material.

Also, the optimization model should incorporate the following economic data:

- capital expenditure and operating costs of IPCC system versus pure truck,
- incremental rate of haulage cost for both systems against length and time estimated [37],
- relocation cost (including the cost for purchasing additional conveyors to increase the length of conveyors),
- the type of conveyors that exit from the pit and their associated costs.

Energy consumption and environmental factors of these two transportation modes are already depicted [38,39]. In this study, these parameters are not in the focus of the authors. Furthermore, the basic assumptions of developed OT–OL model are as follows:

(a) The conventional shovel-truck system is in operation,
(b) The price is constant.
(c) The model should be used for decision-taking about ore OR waste handling individually.
(d) The conventional shovel-truck system will continue to handle the other mining unit (i.e. ore or waste) if the system is applied for one of them.

4.1. The first part: in-pit crusher location optimization

Facility location problems are combinatorial optimization (CO) problems that help strategic management and decision-making. CO is the process of finding the optimal solution for problems within a region of feasible solutions. Location models are generally NP-hard. Furthermore, real-world location problems are often large in scale, and are not solvable to optimality within a reasonable time and effort [40]. The problem of locating facilities is not new to the operations research community; the challenge has inspired a rich, colourful and ever-growing body of literature. A wide range of models and solution approaches with applications ranging across numerous industries have been discussed [41–45].

According to Arabani and Farahani (2012), Facility location problems (FLPs) are divided into two main categories, (1) static facility location problems (SFLPs) and (2) dynamic facility location problems (DFLPs). If the whole of factors and parameters of the problem be fixed and constant through the planning horizon, then the problem is static. But, if the parameters change during the planning horizon and if there is a considerable amount of capital required for facility's development, then the problem is dynamic. Dynamic formulations focus on timing issues involved in locating a facility (or facilities). In these problems, decision-makers must not only select robust locations which will effectively serve changing demand over time, but must also consider the timing of facility expansions and relocations in the long-term. From a general viewpoint, FLPs are sub-divided in terms of two elements: space and time. Continuous-space, discrete-space and network-space location problems are addressed under the category of SFLPs. On the other hand, time-spans constitute the main parts of the DFLPs. DFLPs are sub-divided into the following parts: (1) dynamic deterministic facility location problems, (2) facility location/relocation problems (FLRP), (3) multi-period facility location problems (MPFLP), (4) time-depended facility location problems (TDFLP), (5) stochastic facility location problems which are relatively similar to probabilistic facility location problems and (6) fuzzy facility location problems. It should be noted that some types of DFLPs (especially FLRPS, MPFLP and TDFLP) can be converted to each other (many references considered them as a single model) [46–48].

Facility location models have been used in most of the industries. But these models have rarely been used in the mining industry. In this paper, the in-pit crusher location problem is solved as a location–relocation problem. In FLRP, the decision-maker selects the initial location and the facility's new locations after relocations. Each period varies from the others and has a different condition and due to this difference, the model is a 'dynamic location'. The linear form of dynamic location problem for optimum in-pit crusher location is given in Equations (1).

Objective function:

$$Z = \text{Min} \sum_{k=1}^{r} \sum_{j=1}^{p} \sum_{i=1}^{m_k+1} F_{kij} x_{kij} + \sum_{k=2}^{r} C_k y_k \tag{1.1}$$

Subject to:

$$y_k = 0.5 \sum_{j=1}^{p} w_{kj} \quad \forall k \tag{1.2}$$

$$w_{kj} \geq z_{kj} - z_{(k-1)j} \quad \forall j, k \tag{1.3}$$

$$w_{kj} \geq z_{(k-1)j} - z_{kj} \quad \forall j, k \tag{1.4}$$

$$\sum_{j=1}^{p} z_{kj} = P \quad \forall k \tag{1.5}$$

$$\sum_{j=1}^{p} x_{kij} = 1 \quad \forall i, k \tag{1.6}$$

$$x_{kij} - z_{kj} \leq 0 \quad \forall i, j, k \tag{1.7}$$

$$z_{kj} = \begin{cases} 1 & \text{If in-pit crusher is located in candidate point j} \\ & \text{in period } k, \\ 0 & \text{otherwise} \end{cases} \quad \forall k, j \tag{1.8}$$

$$x_{kij} = \begin{cases} 1 & \text{If face } i \text{ is assigned to condidate point } j \\ & \text{in period } k \\ 0 & \text{otherwise} \end{cases} \quad \forall i, j, k \tag{1.9}$$

$$y_k = \begin{cases} 1 & \text{if in-pit crusher is relocated} \\ & \text{in period } k \\ 0 & \text{otherwise} \end{cases} \quad \forall k \tag{1.10}$$

where r is the number of periods, p is the number of candidate locations, m_k is the number of faces in period k, F_{kij} is the total haulage cost from face i to candidate point j in period k. In order to consider the operating and capital costs of the conveyor from candidate point j to the mill in period k, the value 1 added to m_k on the third summation. Lower conveyor costs than trucks cause in-pit crusher to be situated away from the mill or the waste dump site. Then, it is necessary to embed the costs of these parts into in-pit crusher locating process. C_k is the relocation cost including engineering, disassemble, installation, labour, transportation and overhead costs. x_{kij}, z_{kj} and y_k are binary decision variables.

Equation (1.1) is the objective function of the model and it minimizes the total haulage costs. The first part of the objective function is the summation of operating costs and the second part is the relocating cost of IPCC. The first part of the Equation (1.1) refers to a p-median facility location model. It ensures that the location of an in-pit crusher should be within an optimum distance from each working face. Also, it should consider the total amount of material that must be mined from each working face, which is indicated by the mine production plan. In the case of system relocation, Equations (1.2)–(1.4) add the relocation costs into the model. The Equation (1.5) reveals that, P crusher are exactly opened (P is the number of ore or waste crusher needed). Equation (1.6) certifies that each demand node (face) is assigned in each period. Equation (1.7) indicates that open crushers are only allowed to be assigned to faces. Also, constraint (1.8) to (1.10) set binary conditions for the variables.

In Equation (1), F_{kij} is a function of the amount of material to be hauled in each period and the corresponding costs per ton of material in that period. F_{kij} and haulage costs are a function of haulage distance (d), time (t) and the amount of material to be hauled (Equation (2)).

$$F_{kij}(d,t) = T_{kij} \cdot f_{kij}(d,t) \tag{2}$$

where T_{kij} and f_{kij} are the total amount of material and haulage costs per ton of material in period k that is hauled from site i to destination j, respectively.

Prior to applying Equation (2), the haulage cost functions should be estimated using available methods. There are some methods such as O'hara (2005), and data gathering from similar mines that can be used to estimate the truck haulage costs over the mine life [49]. For the conveyor system, similar investigations should be taken from other mine sites or a function should be developed with respect to local parameters.

4.2. The second part: transferring time optimization

In this step, the objective function of the model in Equation (3.1) considers maximization of discounted cash flows. In the model, the decision variable t indicates the year of mine life that haulage system changes from pure truck to IPCC system. Actually, the problem is in the form of selecting a project from n projects that in this case n is the remaining year of mine life. The point is that, while the IPCC system is used in the middle of mine life, its initial investment will not be made at the outset of the project. Also, IPCCs should be used after the payback period of the mining operation. This is due to the fact that it would be unwise to reinvest in a project that has not yet returned its initial investment. Therefore, for a project with a b-year payback period, investment for an IPCC system is not recommended for earlier than that time. This issue is embedded into the model using Equation (3.2). The parameter b varies from mine to mine.

$$Z = \text{Max} \sum_{k=1}^{t} \frac{CF_{\text{truck}}}{(1+d)^k} + \sum_{k=t}^{r} \frac{CF_{\text{IPCC}}}{(1+d)^k} \tag{3.1}$$

s.t.

$$b \le t \le r \tag{3.2}$$

where CF_{truck} and CF_{IPCC} represent the cash flow of pure truck and IPCC systems, respectively. The variable t is the upper bound of the first summation and it means that the pure truck system is going to be used up to the year t. Also, it is the lower bound of the second summation because the IPCC system will be used from the year t to the end. d is discounted rate and k is periods index, $k = 1, 2, \ldots, r$.

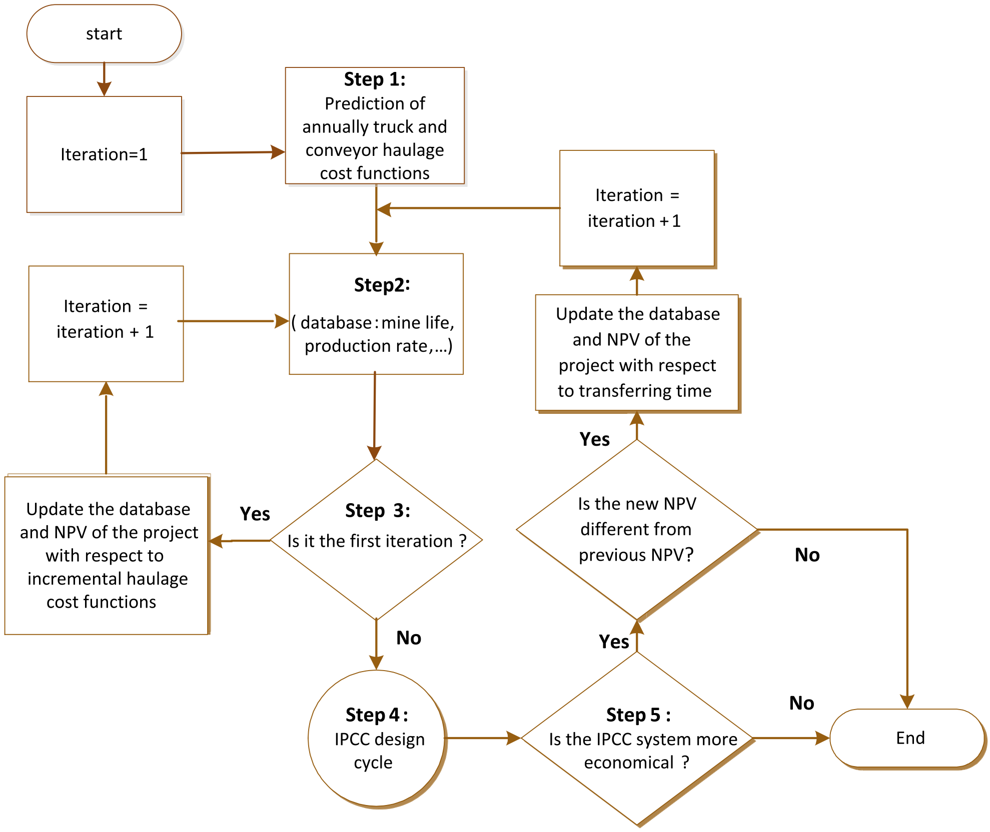

Figure 4. The procedure of finding OT–OL for transferring from pure truck to IPCC system.

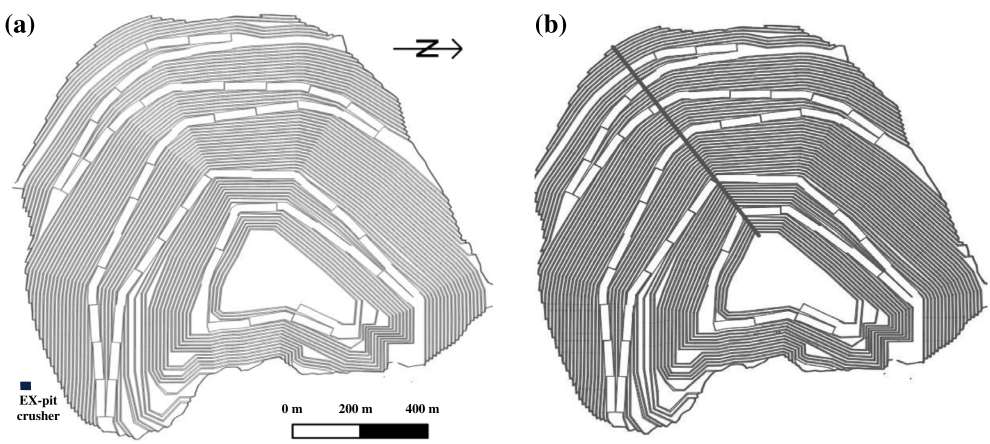

Figure 5. (a) Plan view of pit and ex-pit crusher. (b) Gravity centre line of all levels.

4.3. Optimization procedure

For solving the OT–OL problem a procedure was established wherein the steps are repeated in a circular fashion as further improvements are made. The procedure is presented in Figure 4 and described in

Table 1. Economic and technical parameters of SCM [50].

Economic parameters			Techical parameters		
Parameter	Value	Unit	Parameter	Value	Unit
Operating cost (waste)	1.5	\$/ton-waste	Average bench slope angle	68	degree
Operating cost (ore)	1.68	\$/ton-ore	Ramp width	25–30	metre
Processing cost	1.5	\$/t-ore	Ramp grade	8–10	%
Concentrate transport cost, smelter and refinery cost	800	\$/t-Cu-cathodes	Block size	$25 \times 25 \times 12.5$	metre
Mining administration costs	5,100,000	\$/year	Pit angle	30–40	degree
Processing administration cost	5,100,000	\$/year	First-bench level	2362	metre
Annual discount rate	10	%	Pit-floor level	1600	metre
Copper price	4200	\$/t-Cu-cathodes	Cut-off grade	0.22	%

detail and step-by-step as follows:

Step 1: the annually truck and conveyor haulage cost functions should be predicted using the available methods.

Step 2: technical parameters such as geometry of pit, mine site topography, mining sequences and pit deepening rate should be specified as a database to be used in the next steps. Moreover, it is necessary to assess ore deposit to check its potential for another investment. For long life mining operations, high haulage distances and high production rates (at least 10 Mt/a; and 25 Mt/a is preferred), IPCC systems have more benefits [26, 28 and 29]. Also, it should be answered that transporting which mining unit (i.e. ore, waste or both of them) will be faced with the necessity of transferring to IPCC?

Step 3: if the second step is passed, a question is asked, 'Has the optimum time for application of the IPCC system been set yet?' or in other words, 'is it the first iteration?' If 'yes', using the database, the growth rate of haulage distance and costs over the mine life should be specified. Then, a new final pit limit should be determined using the established haulage cost functions. Decreasing of the volume of the pit with the new haulage cost functions is inevitable [37]. This step will be used in the first iteration, and from the second iteration on, this step will not be considered.

Step 4: in this step the IPCC system will be designed. It includes identifying the conveyor exit method and crusher locations. After identifying the best locations in the whole of mine's life, calculation of incremental haulage cost curve for IPCC will be possible. Indeed, the curve is a parameter used in the next step while identified using OL model's decision variables. For this purpose, dynamic location model in Equation (1) should be applied.

Step 5: in step 4, total haulage cost distributions of the two systems (pure truck and IPCC) is considered that makes it possible to compare the cost of the two systems in the entire mine life. But decision-making with respect to total haulage cost regardless of discount rate and the time value of money will be misleading. At this step, Equations (3) will be optimized. If it is confirmed that the application of IPCC system is economical, then the procedure will continue. Otherwise, the process is ended. If the process is on, after determination of the best time for application of IPCC system and calculation of the new NPV for the project, the new NPV will be compared with the previous one. If the new NPV is better, a new final pit limit with respect to the application of both systems should be calculated and the process should be repeated, otherwise, the process ends.

5. Model verification

To verify the proposed model, the data collected from Sungun Copper Mine (SCM) are used. SCM is a porphyry deposit and is located in the north-west of Iran. It is the second largest copper mine in Iran. Geological reserve of the deposit is estimated up to 806 million tonnes with an average copper grade of 0.62%. The total minable reserve of the deposit is about 388 million tones with an average

Table 2. Annual transportation cost per cubic metre per 500 m [51].

Year	Haulage cost (unit)
1998	26
2000	35
2001	40
2002	46
2003	55
2004	67
2005	90
2006	101
2007	108
2008	127
2009	140
2011	343
2013	332

grade of 0.67% and the stripping ratio is equal to 1.8:1. In the first five years, the annual production amounts to 7 million tonnes and it reaches 14 million tonnes in the remaining years. The mined land area of SCM is 38.2 km^2 and half of it will be completely disturbed during the first 27 years of mine life. Pakhir and Sungun Rivers are streaming through the mine area and they join the Mian-cafe and Ilgene-chai Rivers downstream. The valley of Pakhir in the northern side of the pit is used as a waste dump area. Waste dumps on the same level as mine bench into Pakhir. The economic and technical parameters of SCM which have been used in this study are given in Table 1.

5.1. Step 1

5.1.1. Truck haulage cost function

In this step, truck haulage cost per ton of ore in SCM is calculated. Unit cost can be calculated by dividing the truck hourly cost by the resulting truck productivity. Truck performance is typically expressed in terms of hourly production rate (ton/hour), and calculated considering truck payload and truck total cycle time, which includes spot, load, haul, turn, dump, empty return, wait and delay times [37]. This method requires comprehensive information about each component of haulage cost. In this paper, the annual haulage cost data from 1998 to 2013 is gathered for prediction of truck haulage cost [51]. As shown in Table 2, these data show the haulage costs by truck per 500 m per m^3. To calculate the haulage cost for distances greater than 500 m, these costs are increased linearly. Also, for converting haulage cost from unit per m^3 to the dollar per ton, Sungun ore density and the exchange rate were considered 2.3 tonnes per m^3 and 3000 unit per dollar, respectively.

After normalizing and smoothing the data, the best curve fitted to the data is selected as the model for prediction of haulage cost per m^3 per metre. Finally, the predicted haulage cost function per ton of ore per metre in SCM is calculated as in Equation (4):

$$f(d,t) = 4.0864 \times 10^{-7} \times t^2 \times d - 1.6304 \times 10^{-3} \times t \times d + 1.62637 \times d \qquad (4)$$

where d is truck haulage distance in metres, t is year and $f(d,t)$ is truck haulage cost ($ per ton per metre per year). For example, at a distance of 2 km in year 2030, the haulage cost will be 0.92 $ per ton of ore.

5.1.2. Conveyor haulage cost function

The conveyor is an inseparable part of IPCC system so its design and determination of its specifications are very important. On the other hand, the length of the conveyor and its operating costs affect crusher location. For this purpose, according to the standards of Conveyor Equipment Manufacturers Association (CEMA), conveyor components for carrying ore in SCM are designed. Conveyor operating cost includes spare parts, maintenance, labour and electricity. Calculation of the electricity cost depends on the belt tension and power consumption. Labour, maintenance and spare part costs are considered

Table 3. Distance from each candidate level to each mining level in year 6.

Year 6		Candidate level for in-pit crusher											
		1962	1950	1937	1925	1912	1900	1887	1875	1862	1850	1837	1825
Mining level	1962	575	1570	1488	1469	1482	1792	2155	2431	2769	3189	3062	2948
	1950	1570	575	1148	1128	1142	1452	1815	2090	2429	2849	2722	2607
	1937	1488	1148	580	720	733	1044	1407	1682	2020	2441	2314	2199
	1925	1469	1128	720	580	413	723	1087	1362	1700	2120	1994	1879
	1912	1482	1142	733	413	595	410	773	1049	1387	1807	1680	1566
	1837	3062	2722	2314	1994	1680	1690	1727	1702	1713	1833	585	1265
	1825	2948	2607	2199	1879	1566	1575	1612	1587	1599	1718	1265	615

a percentage of investment cost. After calculation of the conveyor power as a function of conveyor length, incremental annual haulage cost per ton of ore in Sungun is formulated as in Equation (5).

$$f(d,t) = (P_{\mathrm{KW}} \times \mathrm{AOPH} \times 0.03 + (2 \times 2 \times 10^6 \times 12)/3000 + 0.06 \times d \times 3000 \times C_{\mathrm{index}}(t))/14 \times 10^6 \tag{5}$$

where $f(d,t)$ is conveyor annual operating cost per ton per length of conveyor, P_{KW} is power required for transportation of ore to a distance of d metre, AOPH (Annual OPerating Hour) is annual operating hour that in this case is equal to 3600 h, C_{index} (t) is the predicted Marshall and Swift cost index for year t and 14×10^6 is the annual ore production.

5.2. Step 2

Because of valley of Pakhir, waste haulage distance will be varying from 1 to 3 km by the end of mine life, but ore haulage distance increases to more than 5 km. The waste haulage distance is lower than economical travel distance for trucks (less than 2 miles) through the mine life. Therefore, the introduced procedure is implemented for decision-making about transferring of ore haulage system at a certain level of the pit. At the beginning, satisfaction of the remaining mine life and production rate is necessary. Then, assuming that Sungun has a five-year payback period, application of the IPCC system from the sixth year is feasible. At this time, annual production is 14 million tonnes. Thus, Sungun has the potential for application of the IPCC system for ore handling.

5.3. Step 3

5.3.1. Annually haulage distance and cost of pure truck system

The main access road to the pit and the ex-pit crusher are located in the south-east of the mine site at level 1987. It is in the middle of mine depth. Currently, the crusher is located at a distance of 270 m from the southern edge of the pit. After crushing, an 1171-m overland conveyor line moves crushed ore to the concentrate plant in the south-west side of the pit. Figure 5(a) shows a plan view of the pit and ex-pit crusher.

With respect to available mine plan, truck haulage distance (for current truck-shovel system) and the corresponding computed haulage costs over the mine life is determined by Equation (4) (Figure 6).

5.3.2. New ultimate pit limit

To evaluate the reliability of the available mine plan, it is necessary to calculate a new final pit limit with respect to the predicted truck haulage cost (Figure 6). To this end, the average of the predicted haulage costs is added to the other parts of the operating costs (including drilling, blasting, loading and overhead costs). Then, a new final pit limit using the new cost and also the parameter depicted in the Table 1 is determined. But in the case of SCM, the solution doesn't have any noticeable change. It means that the database should not be updated.

Table 4. Optimum locations depending on time for system application.

		Year of mine life considered for application the system																					
	Year	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Opti-	6	1912																					
mum	7	1912	1912																				
levels	8	1912	1912	1912																			
and	9	1912	1912	1912	2025																		
time	10	1975	1975	1975	2025	1975																	
for re-	11	1925	1925	1925	1925	1925	1925																
loca-	12	1925	1925	1925	1925	1925	1925	1912															
tions	13	1887	1887	1887	1887	1887	1887	1887	1887														
of the	14	1825	1825	1825	1825	1825	1825	1825	1825	1825													
sys-	15	1825	1825	1825	1825	1825	1825	1825	1825	1825	1800												
tem	16	1912	1912	1912	1912	1912	1912	1912	1912	1912	1912	1912											
	17	1912	1912	1912	1912	1912	1912	1912	1912	1912	1912	1912	1875										
	18	1862	1862	1862	1862	1862	1862	1862	1862	1862	1862	1862	1875	1862									
	19	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812								
	20	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1812	1775							
	21	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775						
	22	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1775	1750					
	23	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1737	1750	1737				
	24	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725	1725			
	25	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700	1700		
	26	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	
	27	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1650	1637

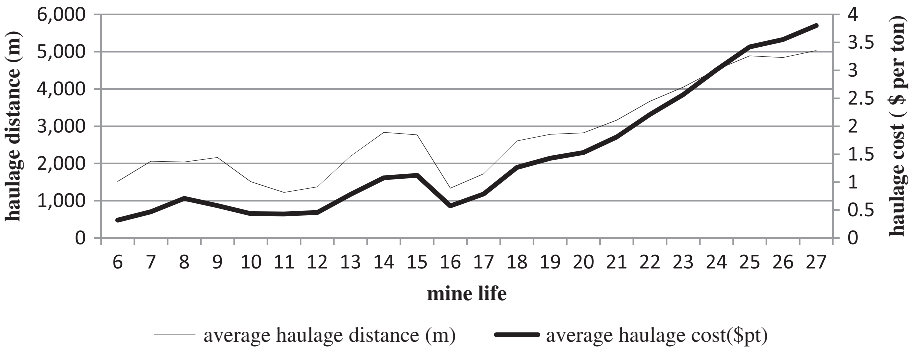

Figure 6. Truck haulage distance and cost over the mine life in SCM.

5.4. Step 4 (IPCC design cycle)

As mentioned, the key point in the IPCC design cycle is the conveyor exit method identification. In this case study, the existing haul roads are selected for routing the conveyor to the pit exit on the level 1987 in the south-east part of mine site. Forty-three thousand tonnes of ore will towed daily and dumped into an in-pit crusher with a capacity of 4000 tonnes per hour. After crushing, the ore is reduced to −250 mm in size and discharged on a discharge conveyor. This belt discharges the crushed ore on1.6 m wide movable conveyor. The conveyor hauls the ore over the existing haul roads out of the pit on to another 1.6 m overland conveyor. The 1171-m overland conveyor moves the crushed ore to the concentrate plant in south-west of the pit. In total, three belts are needed to complete the transport process from pit to mill. As the pit continues to deepen, additional movable conveyors will be installed at a lower elevation to dump on to the first conveyors. The portable crusher and movable conveyors will be relocated to optimize the haulage costs.

5.4.1. Candidate crusher locations

The mine production plan indicates the mining level and the amount of ore material to be mined. To simplify the problem, the gravity centres of each level are assumed as the candidate crusher locations. Figure 5(b) shows the gravity centre line of all mine levels. Furthermore, to improve the solution time, it is assumed that the in-pit crusher can only be placed between the highest and lowest levels of mining in each year. Thus, some penalty values are added to each level outside this boundary. As an example, Table 3 shows the distances from mining levels in each year (i.e. gravity centre of each level) to the candidate point of in-pit crusher location (i.e. gravity centre of each level) in the same year. For the remaining years, these distances are calculated according to the available mine plan.

5.4.2. Solving the OL problem

The dynamic location model (Equations (1)) is adapted to match the case of Sungun. In this case, r = 22 (from year 6 to 27), p = 47 (number of candidate levels) and C_k = \$1.5 million [52]. Then, Equations (4) and (5) are applied to calculate the truck and conveyor haulage costs from each face (level) to candidate levels and from candidate levels to the second crusher, respectively, by the end of mine life. The problem is modelled and solved in GAMS. The model optimizes crusher location and relocation times of the system. Optimum in-pit crusher locations in different years with respect to the time of applying the system are given in Table 4.

In Table 4, the first column shows the years of mine life, and the first row represents the year of mine life when the system is applied. Each number in the table represents the optimum levels for installing the system. As an example, if the system is used from the 10th year, the in-pit crusher should be at level 1975 in the first year and it should be relocated to the level 1925 in the year after. Similarly, the in-pit crusher should be in 12 different levels until the end of mine life and the system should be relocated 11 times.

5.5. Step 5

5.5.1. Haulage cost distributions of pure truck vs. IPCC

After determination of the best location and the best relocations time of the in-pit crusher, accurate comparison of IPCC vs. pure truck systems can be done. Since the time of periodic investments for additional conveyors and crusher relocations are known, calculation of incremental haulage cost for IPCC system (as the one presented in Figure 6 for the pure truck system) is possible. Figure 7 presents the annual IPCC haulage cost over the mine life.

In these calculations, just the inflation of haulage costs are considered and the costs associated with overhead, drilling, blasting and loading operations were assumed to be constant. Distribution of the total operating and investment costs of both systems are shown in Figure 8.

5.5.2. Solving the OT model

After calculation of the cost curves for the two modes of transportation, one can determine the OT. As it is seen from Figure 8, total costs of the two systems are the same up to the 16th year, and after that, costs of the truck system increase more rapidly than IPCC system. The break-even point is

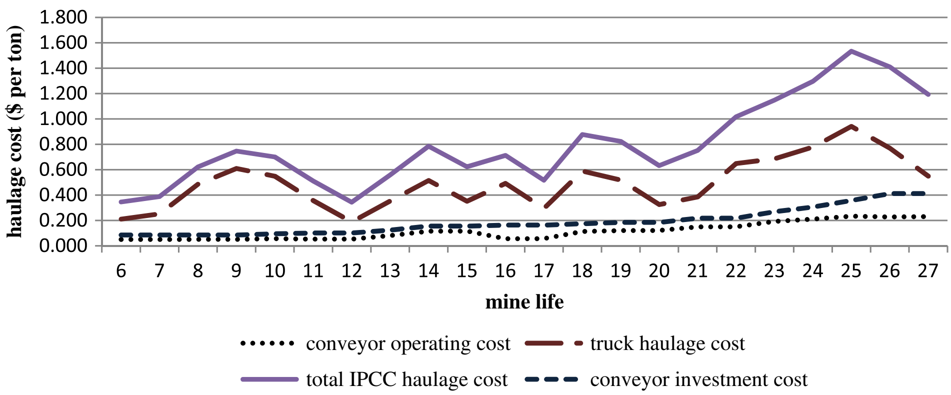

Figure 7. Annually IPCC haulage cost over the mine life in SCM.

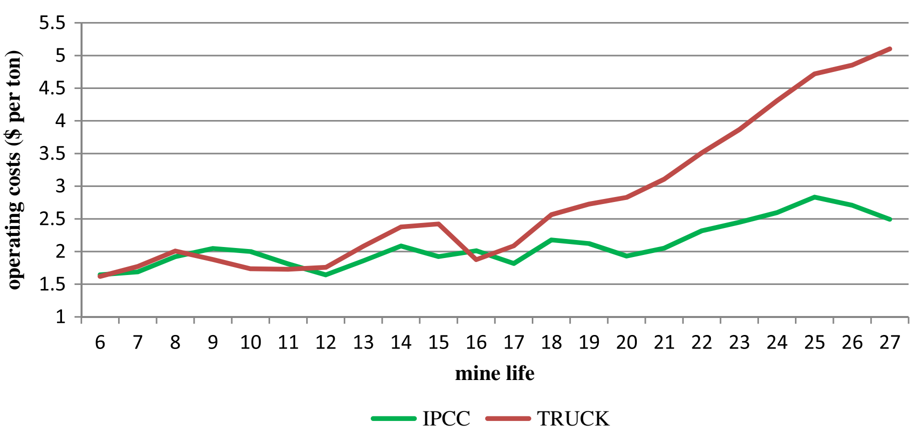

Figure 8. Total annual costs over the mine life for both pure truck and IPCC systems.

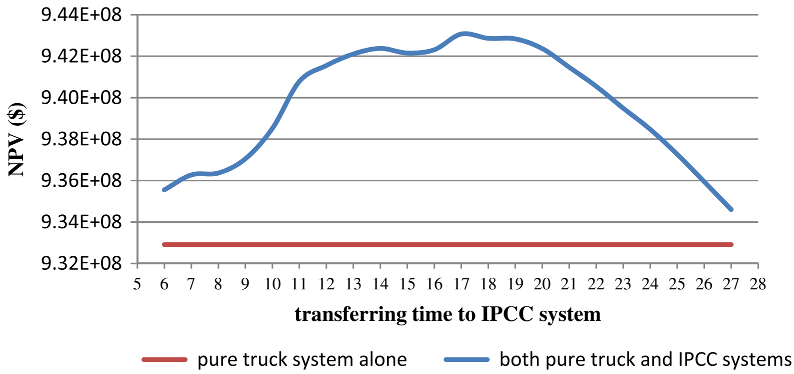

Figure 9. NPV of the project depending on transferring time to IPCC system.

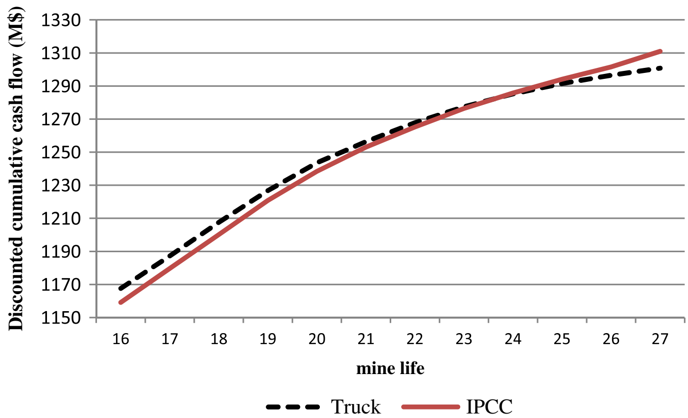

Figure 10. Discounted cumulative cash flow of both system in SCM.

reached within the 16th year of operation. But, this is not optimal since the time value of money is not considered. Therefore, the developed model in Equation (3) is implemented. In this step, you just need to evaluate n options and choose the best one. For this purpose, the data presented in the Table 1 are used. The results show that the optimum time when the haulage system ought to transfer from the shovel-truck system into the IPCC system is the 17th year of mine life (Figure 9). As in Figure 9, the NPV of the project doesn't change noticeably if the IPCC system is used during the years 13–20. It means that this seven-year period is the best time for applying the IPCC system. The blue and red lines show the NPV of the project since both pure truck and IPCC systems and pure truck system alone are used, respectively, by the end of mine life.

5.5.3. Procedure repetition

As presented in the previous chapter, the method will not be run once for all. It is a repetitive sequence of some steps. The procedure should be iterated while further improvements are made. Therefore, a new final pit limit is firstly calculated using the OT and OL models results. While the operating costs are changed, but there was not any perceptible change in the pit outline. It means that the final results are generated in the first iteration.

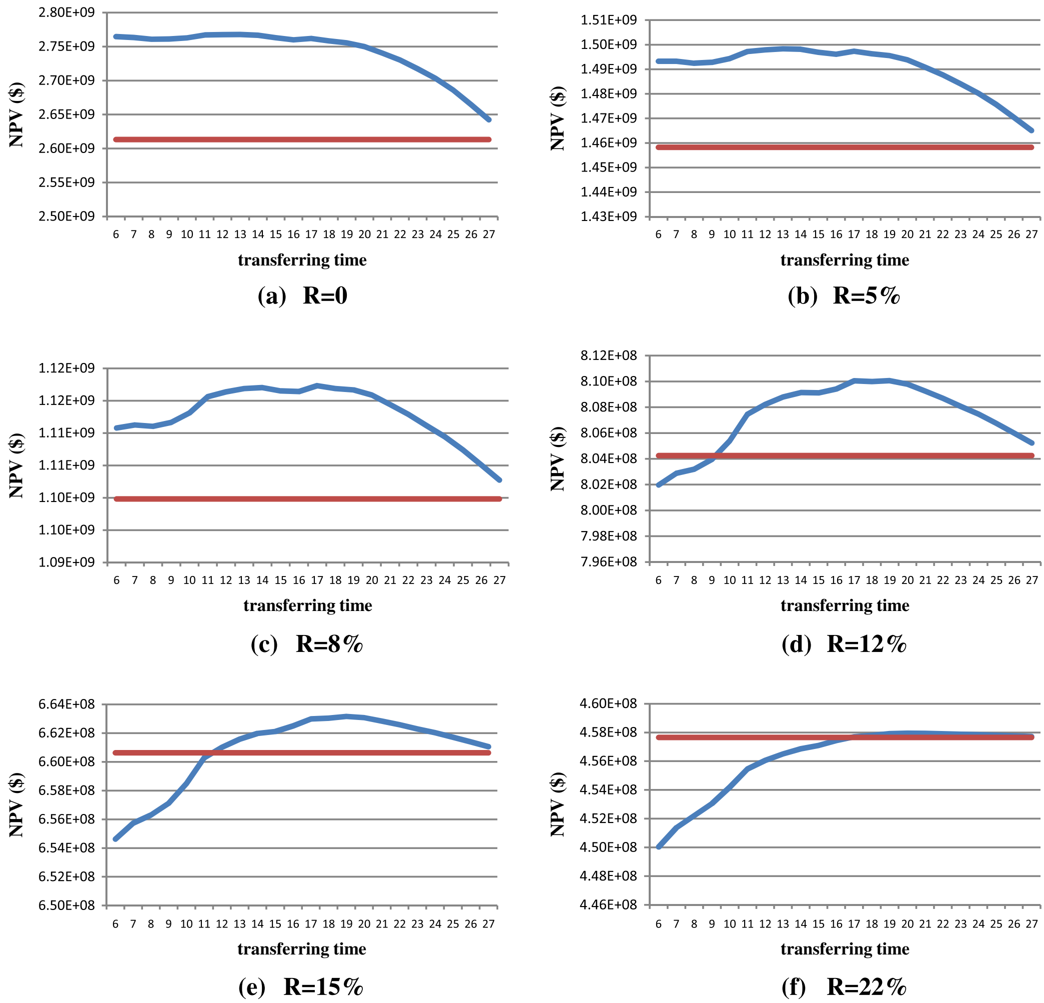

Figure 11. Sensitivity analysis between best transferring time and discount rate.

6. Discussion

The block value can be determined when the haulage system is known. After changing the transportation method, the haulage costs change. Then, the block values change. It would be inevitable that the pit outline gets larger and the system allows low-grade parts to be mined and mining sequences are changed. Determining OT–OL of IPCC system depends upon the pit outline and the mining sequences. The OT problem also depends on the OL problem results. It indicates that the optimum solution for the problem (OL–OT) deals with many interdependent variables and the problem should be solved hierarchically using heuristic trial and error technique. To cope with this, the problem was broken into sub-problems (OT and OL problems).

At the first step, we had some assumption about the time of applying the IPCC system. It is assumed that after payback period, each year of mine life is the time of applying the system. The time helps us know how long should the crusher be used. Without this time, the first mathematical model (OL model) which determines the best locations of the in-pit crusher through the mine life cannot be run. After solving the OL model, the decision-maker should refer back and solve the OT model using the OL model results. Then, he/she should refer to the OL model results and determines the best locations based on the OT model results. For example, in the case of Sungun, assuming different applying time, the OL model was solved. After that, depend on the applying time, the yearly IPCC system costs are calculated. Then, after solving the OT model, the decision-maker refers to the Table 4. According to the Table 4, the level 1875 is the best location for the in-pit crusher in the 17th year. The crusher would be in seven different levels to the end of mine life and the system would be relocated six times.

The heuristic algorithm proposed in this paper is the first try in this regard. The procedure includes a strong logic. Because it is solved mathematically using two models which are tested using the actual data. The models yield the optimum solutions individually. They are linked to each other via an optimization procedure which repeats the optimization till more improvements would not be achievable. But, it cannot be claimed that the solutions are exactly optimal considering that the model works in a discrete time horizon and that the solution method is heuristic. However, due to the interaction between the two sub-problems, the method really works and produces a good quality solution.

The model has improved the NPV of SCM, but the improvement is not significant (Figure 10). The lower improvement of the NPV is more relevant to the number of years that the cash flows are discounted. The cash flow of the last years has lower effect in total NPV. For instance, in the case of SCM, when the system is applied from the 17th year, the cash flows from 17th year to the end are discounted which have a lower effect in NPV. While the total cash flow of the project increased about \$150 million than pure truck mining alone. Also, this is because of the first crusher location in the truck-shovel system. The topography of the mine site permits to place the first crusher in the middle of the pit. This issue causes a lower growth rate of haulage cost in the conventional truck-shovel system.

Further numerical studies are done by modifying the discount rate and relocation costs, to show the different aspects of the method. It seems that relocating the IPCC system each year is impractical and a higher relocation cost should avoid that. Then, the sensitivity analysis was performed on relocation costs. No significant change in the optimum locations occurred by increasing the value of this parameter up to \$ 5 million.

A simulation-based programme was written in Microsoft Excel to show how the optimum applying time changes by varying the discount rate. The results are illustrated in Figure 11. Six different scenarios for discount rate are considered. As shown in Figure 10, it seems that the system is better to be used as soon as possible in a riskless environment (case (a)). The best transferring time moves forward gradually by increasing the discount rate. Simultaneously, the primary years are going to be unfeasible when the discount rate grows (case (d)). This trend continuous until the discount rate grows over 22%. For a discount rate more than 22%, the model doesn't have any solution (case (f)). In another word, for unstable economic condition countries where the interest rate is very high the model probably does not create a solution. The legend of the Figure 11 is as same as the Figure 9.

The production capacity of the SCM is less than the capacity of the available crushers. In some cases where more than one crusher is required, the location model is converted to a p-median instead of the 1-median model. *P* represents the number of in-pit crushers needed. In this way, the model would be able to allocate the material to the proper crusher. Also, application of the IPCC system for transporting both ore and waste can be evaluated individually at the same time while the extraction sequences of ore and waste blocks are the main input of the location model, respectively. In the case of waste, the timing model should handle the extra cost for crushing waste material.

When the existing haul roads are used to handle the materials, it would need to have extra width to accommodate both truck and conveyor haulage lanes. This issue can change the geometry of the pit and also imposes a more extra cost to the project. Furthermore, transferring points and conveyor crossover points should be established. The changes and related costs are not considered in the paper. In the case of SCM, the system is designed in such a way that three transferring points are desired. Conveyor ramp slots require a pushback modification to accommodate the system with a significant anticipation of waste mining from future pushbacks. This is costly and difficult to develop and tunnel option may also be considered if geometrically and economically supportable. The design of the IPCC system should be in such a way that several philosophies of phase/pushback design and also mining flexibility are maintained such that blending constraints can be achieved.

7. Conclusion

Haulage as one main stage of open-pit mining cycle is always done by trucks, especially at the start of the mining operation. In most cases, this stage accounts for half of the mining costs. As time goes by and as the pit becomes deeper, the share of haulage cost increases. This rise will be acceptable up to a threshold. It is the time/location when/where the haulage system should be changed to an IPCC system. This paper presents a methodology to decide the suitable time and location of an IPCC system inside open-pit mines. In fact, the paper proposes an algorithm to identify cost-effective solutions for an important practical problem, the authors did not fancy a problem. To verify the proposed models, they are applied to determine the OT–OL of the in-pit crushing and conveying system in Sungun copper mine of Iran. Additionally, two new mathematical functions are developed for prediction of the annual truck and conveyor haulage cost per ton per distance in SCM. As a result, the 17th year of the mine life is selected as the time for IPCC's application. According to the model, the location for in-pit crusher is recommended on level 1875 which is at 487 m below the highest bench of the pit. In conclusion, a 1% NPV improvement and $150 million cash flow improvement were observed by applying the system in 17th year. It is worth noting that, in some cases, the mining engineers may be faced with the constraining availability of candidate crusher locations over the mine life. Future research should also be focused on dealing with both mine life and production uncertainty in the problem.

Disclosure statement

No potential conflict of interest was reported by the authors.

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