



Determination of the optimal transition point between a truck and shovel system and a semi-mobile in-pit crushing and conveying system

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Synopsis

One of the most challenging aspects in semi-mobile in-pit crushing and conveying (SMIPCC) system design is determining the optimum depth at which to change from a purely truck-based haulage system to a conveyor-based haulage system. We used scenario analysis to determine the optimum transition depth between a truck and shovel (TS) system and a SMIPCC system. Traditional pit-limit algorithms were used to generate the final pit limit on a copper deposit, which was then divided into four pushbacks. The final operating pushbacks (phases) were designed for both TS and SMIPCC. The end depths for each phase are viewed as candidate transition points to switch from the TS to SMIPCC haulage system. Economic calculations were applied for five different scenarios, including adopting SMIPCC from the outset (pure SMIPCC), after the first, second, and third phases, and finally not using the SMIPCC system (pure TS) at all. The analysis indicates that the second scenario, at a depth of 335 m, results in the lowest cumulative discounted cost (CDC). In this case, the CDC is 17.6% lower than that for the pure TS scenario and 10.7% lower than for the pure SMIPCC system scenario.

Keywords

Open-pit mining, mining transportation systems, in-pit crushing and conveying, truck and shovel, transition point.

Introduction

Mining is considered a cost-intensive industry that will yield a profit when the revenue from selling the valuable product exceeds the cost of producing it across the mine life. For maximum profit, designers are looking to increase precision and the ability to optimize production processes throughout the life of mine (Samavati *et al.*, 2018). One way to enhance profits from mining is to identify expensive production processes and provide operational solutions to reduce the cost of these processes.

Transportation costs have always been a significant part of operating costs in large open-pit mines. This is illustrated in Figure 1, which shows a typical operating cost distribution for a large, deep open-pit mine using the conventional truck and shovel (TS) system. Transportation costs are very variable, depending on pit configuration and geographical location. However, the haulage component is often about 45% of operating costs on a life of mine basis (Tutton and Streck, 2009). As the pit depth increases, the greater the distance and cycle time for trucks hauling material out of the pit. As a result, more trucks are required to transport a specified volume of material. Fluctuations in fuel, tyre, and spare parts prices, and greenhouse gas emissions due to the truck and shovel system, may increase operating costs and cause environmental impacts.

Due to the depletion of many high-grade and near-surface mineral sources, mining operations have expanded to exploit mostly low-grade and deeper deposits (Osanloo, 2012). Thus, it is necessary to use methods that entail lower extraction costs in addition to greater environmental compatibility so that low-grade and deep mineral resources can be extracted viably.

The concept of using a conveyor belt to transport material from the pit was first mooted at an open pit mine in Germany in 1956, due to the wet and soft ground conditions which made it difficult to use trucks (Koehler, 2003; Utley, 2011). Due to the limited range of materials sizes that could be transported by conveyors (Terezopoulos, 1988), a mobile in-pit crusher system was used for crushing the extracted material. The in-pit crushing and conveying (IPCC) system is a combination of these two types of equipment (conveyor and crusher). Although this system was first used because of poor road conditions, today, advances in the design and construction of conveyors receive more attention. IPCC

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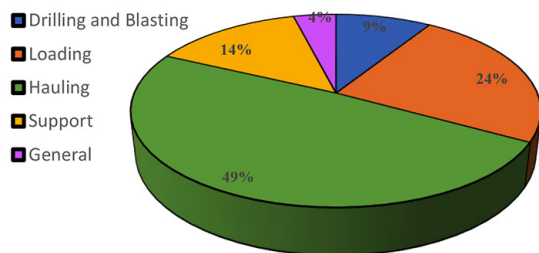


Figure 1—Typical operating costs distribution in large open-pit mines throughout the life of mine, based on a 600 kt/d operation examples in Chile using the TS system (Tutton and Streck, 2009)

systems have lower operating costs than TS systems because of their continuous operating regime, reduced labour requirements, and lower energy consumption. However, they require a higher capital cost and are less flexible (Nehring *et al.*, 2018).

In conventional TS mining the extracted materials are loaded into trucks after blasting. Waste materials are sent to a waste dump while ore is sent for primary crushing before being stored on a run-of-mine (ROM) pad prior to feeding into the processing plant. In this method, the crusher is located outside the final pit limit. Due to the short transportation distance in the initial years of mine life, the haulage cycle time is short. However, as the mine life matures and the depth of the mine increases, the distance that material needs to be transported increases. This causes the truck cycle time to lengthen. As shown in Figure 2, the incremental increase in haulage distance reduces the hourly throughput of trucks. This generally results in the need for more trucks, and in turn a significant capital reinvestment. It also increases operating costs. Accordingly, reducing the truck haulage distance may be a good method for decreasing haulage costs (on a tons per hour basis).

In the IPCC haulage system, extracted material is transported from within the pit by conveyor belts. To do so, materials are initially crushed by an in-pit crusher to a size range that allows efficient transportation to their destination. If the extracted material feeds into conveyors after passing through the crusher, the haulage system is termed a fully-mobile in-pit crushing and conveying (FMIPCC) system. Similarly, if the conveyor belt is fed by trucks, the haulage system is a semi-mobile in-pit crushing and conveying (SMIPCC) system (Frizzell and Martin, 1992). In the SMIPCC system, the transfer of waste or ore from the upper and lower benches to the in-pit crusher is carried out by trucks. This system combines a continuous (conveyor belt) and discontinuous (truck) system and has the advantages of both systems (Paricheh, Osanloo, and Rahmanpour, 2017). In the case of a breakdown in the system (trucks or conveyors), the transportation process may often continue, albeit at a reduced capacity. The SMIPCC system is therefore viewed as a lower risk system and is often thus preferred over FMIPCC, which is a purely continuous system.

Because the transportation route is shorter, a smaller fleet of trucks is required; however, the number of loading machines is the same as for the TS system. To avoid increasing the haulage distance, the crusher(s) location may be changed at regular intervals (Rahmanpour *et al.*, 2014). Using this method can save costs and improve the economics of the operation (Kochanowsky, 1961; Terezopoulos, 1988; Zimmermann and Kruse, 2006; Szalanski, 2010; Dean *et al.*, 2015; Nehring *et al.*, 2018; Paricheh and Osanloo, 2019a; Nunes *et al.*, 2019; Hay *et al.*, 2020). Reduced fuel requirements, energy consumption,

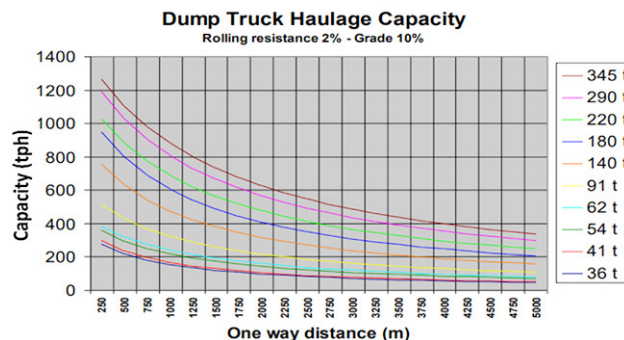


Figure 2—Effect of transportation distance on hourly throughput of trucks with different capacities (Zimmermann and Kruse, 2006)

and pollutant gas emissions are further benefits of this system (Norgate and Haque, 2013; Purhamadani, Bagherpour, and Tudeszki, 2021).

Numerous researchers have investigated use of an in-pit crusher (Hays, 1983; Huss, 1983), conveyors (Kesimal, 1997; Paricheh and Osanloo, 2019b), and high-angle conveyor (dos Santos, 1984; Mitchell and Albertson, 1985; dos Santos and Staniscic, 1986; dos Santos, 2016; Liu and Pourrahimian, 2021). Others have addressed the problem of optimally locating the crusher (Tudeszki *et al.*, 2004; Konak, Onur, and Karakus, 2007; Roumpou *et al.*, 2014; Paricheh, Osanloo, and Rahmanpour, 2017, 2018; Abbaspour *et al.*, 2019; Paricheh and Osanloo, 2019a, 2019c), and economic advantages and disadvantages. Nevertheless, insufficient research has been done concerning the optimal transition point (time or depth) between a TS system and SMIPCC.

In open-pit mining operations in most developing countries, the material transportation cycle is mainly discontinuous and carried out using the conventional TS system. Due to the superiority of IPCC, if one wants to use this system, the first major question is when or at what depth to shift from the TS to the IPCC system. In this paper we present an innovative approach whereby the end of each pushback/phase is considered as a potential point in the operation to switch the haulage system. Considering five potential transition point scenarios, economic calculations (determining operating and capital costs) are performed for both systems.

Subject modelling

Six main steps have been defined and carried out in this study to achieve accurate and consistent comparisons. Figure 3 shows a flow chart of the various aspects of the process used to generate and evaluate the use of IPCC as part of the mine planning process.

- Step 1: Geological and exploratory review to determine and assess the deposit location and the topography of the area, volume, tonnage, grade, density, and grade-tonnage relationship.
- Step 2: With these new estimations, the possibility of open pit mining is investigated. If the conditions for open pit mining are favourable, the final pit limits and extraction scheduling are then determined. In the case of unfavourable conditions for open pit mining, underground mining methods should be evaluated.
- Step 3: After defining the final pit limit and extraction sequence, operating costs are calculated. According to

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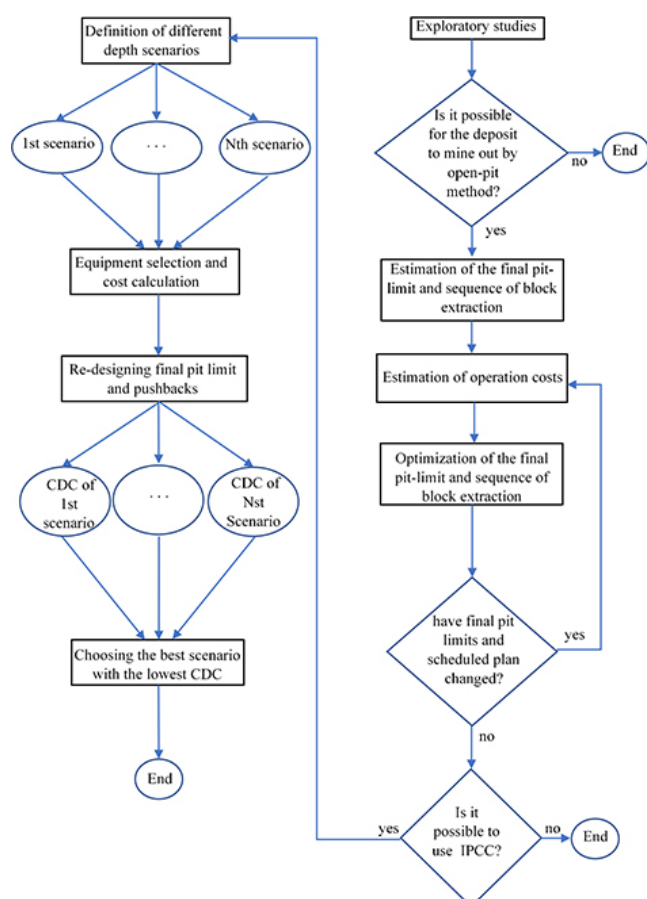


Figure 3—Flow chart of investigation for the implementation of IPCC

operational expenses, the final pit limit and schedule may change to achieve the most optimal plan.

- Step 4: At this stage, the utilization of the IPCC system is subjected to a detailed feasibility study. Influencing factors in decision-making include production rate, mine life, topography, weather conditions, environmental laws, access to fuel and energy resources, availability of required machinery, strength and hardness of the rock, *etc.* Based on these factors, if the implementation of IPCC is not feasible for any reason, the decision part of the flow chart will end. Otherwise, different scenarios are determined to transition the haulage system to IPCC as the depth increases.
- Step 5: At this stage, research is carried out on the equipment for each haulage system (conveyor belt type, width, slope, belt count, and cost of the conveyor belt system, crusher cost, cost of trucks, and truck count). Furthermore, each pushback and final pit limit are re-designed according to conveyor exit restrictions and crusher location.
- Step 6: According to the cost parameters, the CDC of each scenario is calculated. The lowest CDC option should be selected as appropriate to change the method.

Assumptions

A number of assumptions were made in this research as follows:

1. Crushing costs are the same for both in-pit (in IPCC system) and out-of-pit crushers (in the TS system).
2. The same system (TS or SMIPCC) is used for both waste

and ore.

3. There is no mixing of ore and waste materials as a result of using the SMIPCC system. In this case, there are separate crusher and conveyor systems for waste and for ore handling.
4. There is only one crusher for ore and one crusher for waste in the SMIPCC system. In the TS system, there is only one fixed ore crusher outside of the final pit limit, while waste material does not need crushing and is delivered to the waste dump directly by truck.
5. Uncertainty related to all parameters such as operating costs (electricity, fuel, crusher movement, *etc.*) and capital costs (conveyor, crusher, truck, and spreader) has not been considered.
6. The same mathematical final pit limit (not the operating final pit limit) in both TS and SMIPCC systems is assumed. In this case, there is no significant difference between the final pit limits of the SMIPCC and TS systems.
7. The tonnage-grade distribution of the deposit is the same across both systems. Therefore, when considering a constant ore price, the income per ton of ore is the same in both systems. As such, the economic investigation is carried out based on cumulative discounted operating and capital costs.

Case study

As a case study, a conceptual cylinder-shaped copper deposit has been used. The average radius and depth of the orebody are 345 m and 670 m respectively, with 20 m of overburden. A block model of the orebody was initially constructed, and nested pits were generated based on different product prices. Technical and economic information that forms the basis of this investigation is provided in Table I. Figure 4 describes the amount of ore, waste, and stripping ratio located within the final pit limit for different copper prices. As the copper price increases, the amounts of ore and waste inside the final pit limit increase. However, the stripping ratio may increase or decrease, depending on (1) increasing the rate of new ore due to conversion of waste to ore as the price increases and cut-off grade decreases; (2) increases in new waste when the final pit is extended and more overburden is required to be removed.

A minimum working bench width of 80 m is applied, which in this case generates four pushbacks to fully exploit the pit limit. Figures 5 shows a schematic view of the pushbacks and final pit

Table I

Parameters required for estimating the reserve and designing of the final pit

Parameter	Amount (unit)
Ore density	2.6 (t/m ³)
Waste density	2.3 (t/m ³)
Copper price	5 900 US\$/t
Selling cost	356 (US\$/t)
Mining cost	2 (US\$/t)
Stripping cost	1.8 (US\$/t)
Dilution	5%
Mining recovery	95%
Milling and flotation cost	7.0 (US\$/t)
Processing recovery	85%
Maximum stable pit slope	45 (degrees)
Minimum pit floor width	50 (m)

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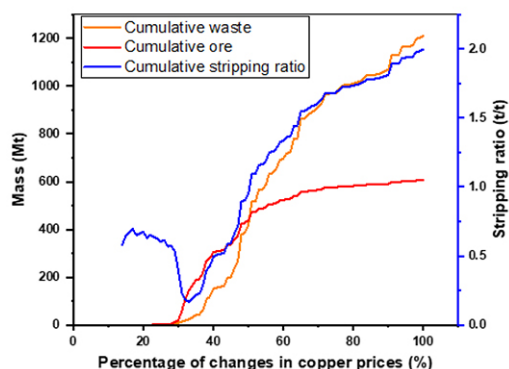


Figure 4—Stripping ratio along with ore and waste extraction in the nested pits for various prices of copper

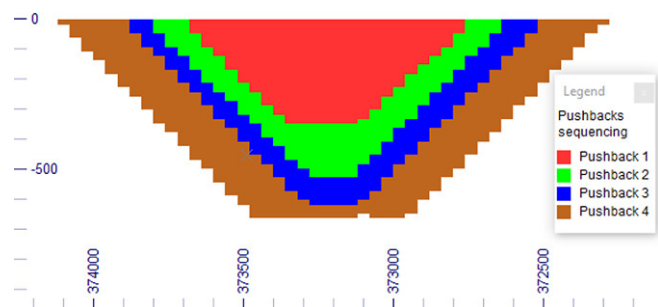


Figure 5—East-west section of pushback sequencing

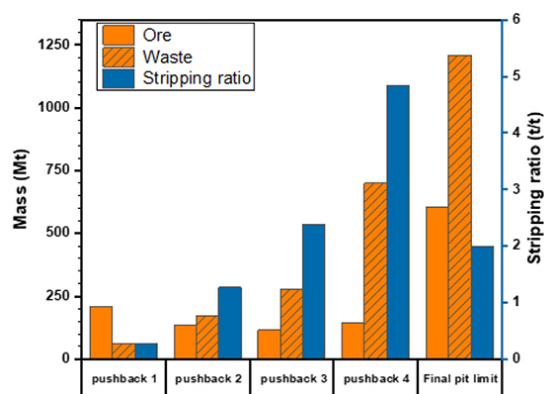


Figure 6—Ore and waste in each pushback and final pit

limit, which is determined based on maximizing profit. Figure 6 shows the ore, waste, and stripping ratio associated with each pushback and final pit limit.

After determining the pushbacks, the mine production capacity of 20 Mt per year was derived using Taylor's equation. The final depth of each pushback is considered a potential transition point of the haulage system from TS to IPCC. The

centre of gravity of ore and waste for each pushback was used to best locate the in-pit crusher.

Five scenarios are considered in total, as follows:

1. SMIPCC used from the outset (SMIPCC only)
2. Transition to SMIPCC after pushback 1
3. Transition to SMIPCC after pushback 2
4. Transition to SMIPCC after pushback 3
5. SMIPCC not used (TS only).

The operating pushbacks (phases) and final pit limit were designed for both systems with a bench height of 15 m and a face slope angle of 65 degrees. Figures 7 to 10 show the plan view of four pushbacks using TS and SMIPCC.

The final operating pit limit of scenarios 1 to 4 is the same, with the amount of waste of these four scenarios increasing by approximately 90 Mt compared to the fifth scenario (pure TS), and 100 Mt compared to the optimal final pit shell. This increase is due to the additional waste generated during construction of new ramps for the conveyor path and switchbacks in the truck ramp to avoid intersecting the conveyor and truck roads. Details of the increasing waste in each designed phase of the two systems (compared to the optimal pit shell) are described in Table II. The total amount of waste in the final pit limit is increased by 7.4% under the SMIPCC scenario compared to TS. However, the amount of ore does not change.

Due to the characteristics of the mine, the need for trucks with high capacity is evident. Caterpillar 793C trucks (Caterpillar, 1998) with a capacity of 221 t were found to be suitable. According to the performance of this truck, the cycle time was calculated for each phase.

The centre of gravity of waste and ore was used to measure transportation time in the TS system along the path length. In the SMIPCC system, four centres of gravity were considered – two for ore blocks above and below the crusher, and two for waste blocks. Transportation time was calculated using the abovementioned centres of gravity and the specific paths for hauling ore and waste from each of them. Table III contains the information used for calculating the efficiency of the TS system.

Using this information, efficiency was calculated at 62% for each shovel and 70% for each truck. Data for calculating the required number of trucks is presented in Table IV.

As shown in Table II, the amount of stripping required increases from the first to the fourth pushback. As such, the number of machines for executing each phase needs to gradually increase. Since ore production capacity has been set at 20 Mt/a, the capacity of the conveyor for ore extraction will remain consistent.

Considering three 8-hour work shifts and 310 working days in a year (55 days for repair, replacement, daily and monthly servicing), the required capacity of the ore conveyer was

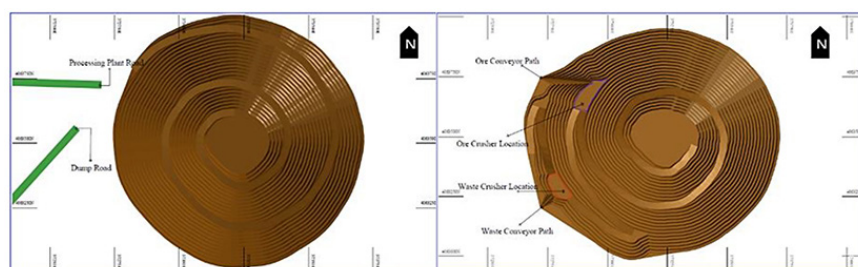


Figure 7—Pit appearance after extraction of first pushback with two haulage systems (left: TS and right: SMIPCC)

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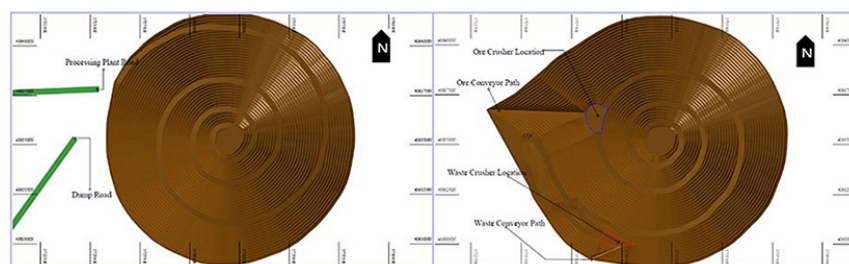


Figure 8—Pit appearance after extraction of second pushback with two haulage systems (left: TS and right: SMIPCC)

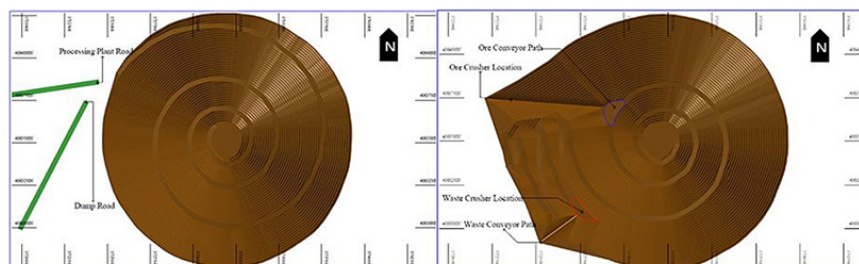


Figure 9—Pit appearance after extraction of third pushback with two haulage systems (left: TS and right: SMIPCC)

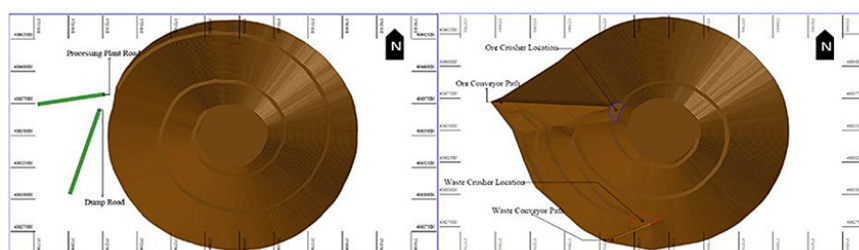


Figure 10—Pit appearance after extraction of fourth pushback with two haulage systems (left: TS and right: SMIPCC)

Table II

Increasing waste in pushbacks for TS and SMIPCC systems compared to optimal pit shell

Increase in waste	TS (Mt)	SMIPCC (Mt)	Difference (Mt)
Phase 1	61.8	71.8	10
Phase 2	205.6	223	17.4
Phase 3	266.8	297.8	31
Phase 4	684.7	716.1	31.4
Final operating pit limit	1218.9	1308.7	89.8

Table III

Information on Idle times of truck and shovel equipment

Cause of delay	Amount
Bad weather	20 days per year
Shift change	0.5 hours per shift
Truck breakdown and repair	15 days per year
Shovel breakdown and repair	30 days per year
Unpredictable idle time	10%

estimated at 2690 t/h. The capacity of the stripping conveyor, however, will differ in each phase. Thus, the equipment required in each phase needs to be computed separately. A brief description of the machines required for each phase is provided in Table V. For example, if it is decided to mine using the third scenario, the machinery for the first two phases will be chosen

Table IV

Information required to calculate the number of needed equipment items

Parameters	Value
Ore density	2.6 t/m ³
Waste density	2.3 t/m ³
Ore load fill factor	0.85
Waste load fill factor	0.9
Delay time factor	0.95
Shovel loading cycle	30 seconds per cycle
Truck waiting time	30 seconds per cycle
Truck manoeuvre time	30 seconds per cycle

from the TS system and for the remaining two phases from the SMIPCC system.

Based on economic analysis of the various scenarios, it is apparent that the purchase of single large waste conveyors and spreaders is preferred over multiple lower capacity sets of equipment. Therefore, the required machinery for each scenario has been selected based upon the lowest CDC. Capital and operating costs of the equipment of both systems are calculated using InfoMine cost tables and are presented in Table VI. Different components of the SMIPCC haulage system for each scenario are presented in Table VII. Economic studies for each scenario are presented in Table VIII. The most economical haulage option is chosen between the TS and SMIPCC alternatives.

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Table V

Machines required for the execution of each phase

System type	Required components	Phase 1	Phase 2	Phase 3	Phase 4
SMIPCC	Sloped waste conveyor length (m)	100	200	312	420
	Sloped waste conveyor capacity (t/h)	900	4150	7650	14 800
	Sloped ore conveyor length (m)	260	590	807	1030
	Horizontal waste conveyor length (m)	870	800	700	700
	Horizontal ore conveyor length	980	580	430	216
	Shovel count	2	3	4	6
	Truck count	5	9	18	29
TS	Shovel count	2	3	4	6
	Truck count	8	21	38	79

Table VI

Haulage equipment capacity with corresponding economic information

Equipment type and capacity	Capital cost (US\$ million)	Annual operating cost (US\$ million)
Truck – 221 t	4.86	2.10
Spreader – 1270 t/h	7.57	3.53
Spreader – 3628 t/h	9.60	4.78
Spreader – 4525 t/h	10.89	5.54
Horizontal conveyor – 454 t/h (1615 m length)	4.50	3.65
Horizontal conveyor – 907 t/h (1615 m length)	5.10	4.32
Horizontal conveyor – 3629 t/h (1615 m length)	8.40	7.72
Sloping conveyor – 454 t/h (610 m length)	1.97	1.56
Sloping conveyor – 907 t/h (610 m length)	22.38	1.96
Sloping conveyor – 3629 t/h (610 m length)	4.30	3.28

Table VII

SMIPCC equipment for each scenario

Phase	Scenario 1		Scenario 2	
	Conveyor belt	Spreader	Conveyor belt	Spreader
1	One 907 t/h belt	One 4535 t/h spreader	-	-
2	One 3629 t/h belt	-	One 3629 t/h and one 907 belt	One 4535 t/h spreader
3	One 3629 t/h belt	One 3629 t/h spreader	One 3629 t/h belt	One 3629 t/h spreader
4	Two 3629 t/h belts	Two 3629 t/h spreaders	Two 3629 t/h belts	Two 3629 t/h spreaders
Phase	Scenario 3		Scenario 4	
	Conveyor belt	Spreader	Conveyor belt	Spreader
1	-	-	-	-
2	-	-	-	-
3	Two 3629 t/h and one 454 t/h belt	One 3629 t/h and one 4535 t/h spreader	-	-
4	Two 3629 t/h belts	Two 3629 t/h spreaders	Four 3629 t/h and one 454 t/h belt	Three 4535 t/h and one 1270 spreader

The sensitivity of the optimal scenario and transition point to discount rate is shown in Figure 11. As indicated, the optimal transition point between TS and SMIPCC systems is not reasonably related to the discounted rate. The second scenario, with a depth of 335 m for transition from TS to SMIPCC, has the lowest CDC across all discount rates. A comparison between pure TS and pure SMIPCC shows that the pure SMIPCC system has a higher economic advantage (lower CDC) than the pure TS system at a zero discounted rate. This advantage continues up to the 10% discounted rate. At a discounted rate of 10% or more, the pure TS system has a lower CDC than pure SMIPCC because the operating costs do not affect the CDC significantly at a high discounted rate in the latter periods of mine life. Therefore, the pure TS system with a lower capital cost than the SMIPCC will be selected as the optimum system. Choosing between pure TS and pure SMIPCC systems is highly sensitive to the discount rate, but the optimum transition point is not.

Conclusion

The SMIPCC system is one of the most appropriate options for mining deeper and lower-grade deposits due to its lower operating costs, which can thus reduce the cut-off grade. However, a greater capital investment is required in comparison to conventional truck and shovel systems. Whether or not the greater initial capital investment can be recouped throughout the mine life from lower operating costs should be the subject of technical and feasibility studies in order to identify the most appropriate system(s) and the point at which a transition may occur.

For the case study that was presented, a depth of 335 m was determined as the optimum transition point from TS to SMIPCC based on a CDC analysis. It was also found that the transition point is not sensitive to the discount rate. However, because the operating costs do not influence the CDC substantially in the

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Table VIII

Summary of economic analysis (costs in million US\$)

Phase	Parameter	Scenario 1	Scenario 2	Scenario 3	Scenario 4	Scenario 5
First	Number of required trucks	5	8	8	8	8
	Capital cost for purchasing trucks	24.30	38.88	38.88	38.88	38.88
	Operational cost of trucks (per year)	10.50	16.80	16.80	16.80	16.80
	Capital cost for purchasing conveying system	10.07	0	0	0	0
	Operational cost of conveying (per year)	10.07	0	0	0	0
	Capital cost for purchasing crusher	20.00	0	0	0	0
	Capital cost for purchasing spreader	10.89	0	0	0	0
	Operational cost of spreader (per year)	5.45	0	0	0	0
	Additional stripping costs (per year)	0.37	0	0	0	0
	Transportation SMIPCC cost at the end of phase	2.00	0	0	0	0
Second	Number of required trucks	4	1	13	13	13
	Capital cost for purchasing trucks	19.44	4.86	63.18	63.18	63.18
	Operational cost of trucks (per year)	18.90	18.90	44.10	44.10	44.10
	Capital cost for purchasing conveying system	8.29	17.46	0	0	0
	Operational cost of conveying (per year)	14.51	14.32	0	0	0
	Capital cost for purchasing crusher	0	20.00	0	0	0
	Capital cost for purchasing spreader	0	10.89	0	0	0
	Operational cost of spreader (per year)	5.45	5.45	0	0	0
	Additional stripping costs (per year)	0.97	0.97	0	0	0
	Transportation SMIPCC cost at the end of phase	2.00	2.00	0	0	0
Third	Number of required trucks	9	9	0	17	17
	Capital cost for purchasing trucks	43.74	43.74	0	82.62	82.62
	Operational cost of trucks (per year)	37.80	37.80	37.80	79.80	79.80
	Capital cost for purchasing conveyor system	8.60	8.60	22.49	0	0
	Operational cost of conveying (per year)	21.52	21.33	20.03	0	0
	Capital cost for purchasing crusher	10.00	10.00	30.00	0	0
	Capital cost for purchasing spreader	9.60	9.60	20.49	0	0
	Operational cost of spreader (per year)	10.23	10.23	10.23	0	0
	Additional stripping costs (per year)	2.38	2.42	2.42	0	0
	Transportation SMIPCC cost at the end of phase	3.00	3.00	3.00	0	0
Fourth	Number of required trucks	11	11	8	0	41
	Capital cost for purchasing trucks	53.46	53.46	38.88	0	199.26
	Operational cost of trucks (per year)	60.90	60.90	60.90	60.90	165.90
	Capital cost for purchasing conveyor system	16.72	13.08	16.65	38.10	0
	Operational cost of conveying (per year)	35.55	32.02	33.99	34.05	0
	Capital cost for purchasing crusher	20.00	20.00	20.00	50.00	0
	Capital cost for purchasing spreader	19.20	19.20	16.80	40.24	0
	Operational cost of spreader (per year)	19.78	19.78	19.78	19.89	0
	Additional stripping costs (per year)	1.94	1.94	1.94	1.94	0
	Transportation SMIPCC cost at the end of phase	–	–	–	–	–
CDC (considering a discounted rate of 5%)		837.51	747.83	765.70	798.17	907.22

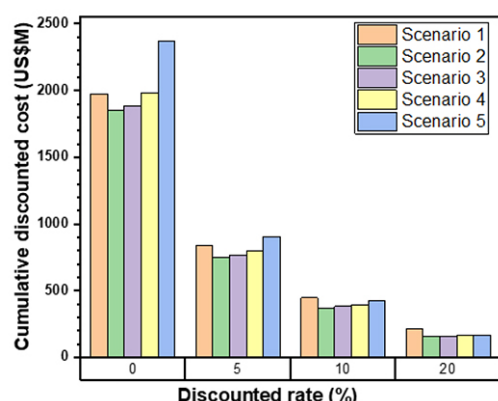


Figure 11—Cumulative discounted cost for each scenario at various discount rates

latter periods of mine life, with a high discount rate the pure TS system, with a lower capital cost, is the optimum when deciding between pure TS and pure SMIPCC systems. Conversely, at a lower discounted rate pure SMIPCC is more economically viable than the TS system.

Future work

Future research should investigate the uncertainty related to equipment operating and capital costs. This should perhaps be combined with determining the optimum ultimate pit limit and mine plan based on mathematical modelling. Further work addressing the optimal transition point between a TS system and a SMIPCC system relating to the optimum location and relocation of the semi-mobile crusher should also be considered. Even though fully mobile in-pit crusher conveyor (FMIPCC) systems require more extensive redesign of the mining operation due to the introduction of sequencing constraints, there are further economic benefits that could result from such a system which could also be addressed.

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Determination of the optimal transition point between a truck and shovel system

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