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Technical and economic assessment of dry stack tailings alternatives for an iron ore project: a case study

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Abstract

With a combination of highly publicised tailings dam failures and acute water shortages in some locations around the world, mining projects are facing more stringent requirements relating to safety, sustainability and environmental acceptance. This paper focuses on a conceptual study related to a dry stack tailings (DST) project for an iron ore operation in North America. The study consisted of two parts: firstly, the effect material properties have on the process selection and deposition of the tailings, and secondly, the dewatering, transport and stacking alternatives (taking into account the topography of the permitted tailings storage located in a mountainous area). The case study also included the overall investment and operating cost estimation for the storage of approximately 70 Mt of tailings over a period of nine years. The paper describes, step by step, evaluation of the key input parameters, the development of a time usage model in connection with the selected operating schemes, and the equipment required for the processing, transport and stacking of the tailings.

Keywords: *material testing, operating scheme options, time usage model, equipment selection and sizing*

1 Introduction and project summary

TAKRAF Group was commissioned in October 2019 by Knight Piésold as the main contractor to perform a conceptual study investigating the production and handling of filtered iron ore tailings, inclusive of stacking into a preferred dump area. The main parts of the study are the determination of technical specifications and the required material testing for thickening and filtration, and the assessment of possible and feasible transport options, dumping methods and technologies, inclusive of investment and operating cost estimations.

In the first phase, assessment of the expected tailings capacity, testing of the material (drying tests) and sizing of the dewatering plant was undertaken by a team from TAKRAF Canada in Vancouver, which provided basic data for the layout of the dry tailings conveying and dumping system.

In the second phase, required key input parameters (e.g. material data for conveying and dumping), a time usage model, and operating and economic parameters were defined. After discussion of the proposed options of dump bodies placed within the given tailings storage facility (TSF) border, possible transport and dumping alternatives were discussed and evaluated. In parallel, Knight Piésold carried out the important geotechnical assessment related to a large multi-lift tailings dump.

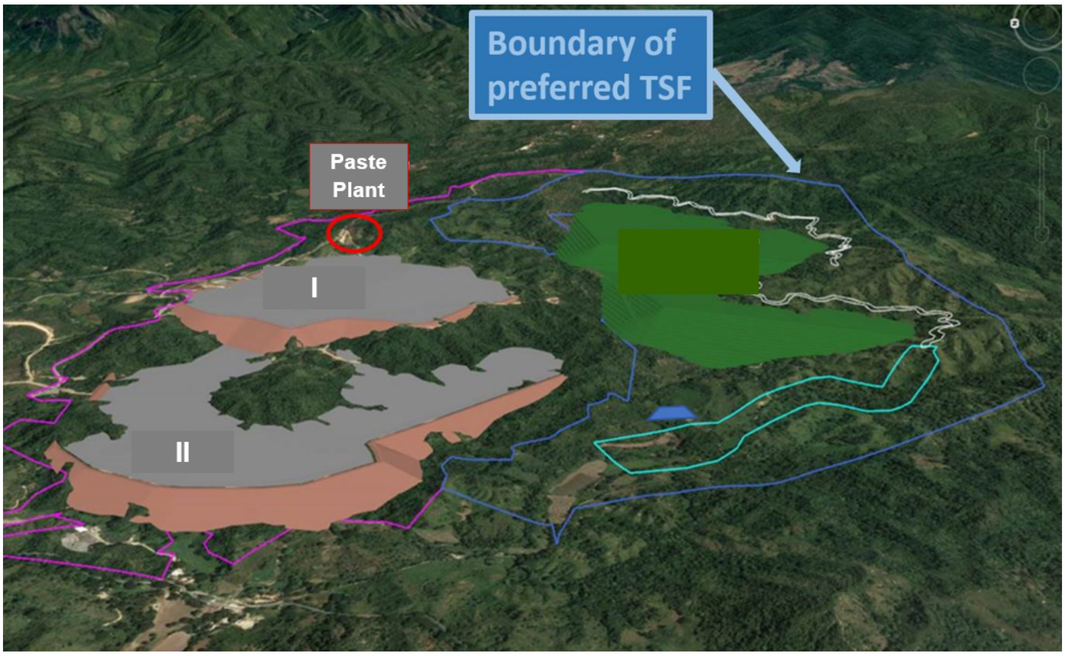


Figure 2 Site location of existing tailings impoundments (I and II) and the preferred tailings storage facility. Source: client's project document

2 Material testing

The objective of the material testing phase was to determine the best dewatering process in terms of operational expenditure (opex) and capital expenditure (capex) for the current total tailings and future tailings.

The analysis of the tailings types was focused on general geotechnical features and dewatering properties to provide the database for the subsequent conceptual design phase.

2.1 Test plan for tailings filtration

The testing phase involved a site visit to discuss the sampling points and the TSF. Two separate composite samples were prepared for testing, current and future tailings (Figure 3). After sample characterisation (Figure 4), the settling and thickening tests were performed to evaluate the process parameters (Figure 5) to be applied to the existing two 34 m diameter DELKOR paste thickeners. These parameters were then used to produce underflow slurries for filtration testing (Figure 6). Initial filtration tests were done to select the equipment suitable for this application. After optimising the filtration parameters for both current and future tailings (Figure 3), 40 kg of filter cake samples were generated for geotechnical testing from tailings with higher cake moisture (worst-case).

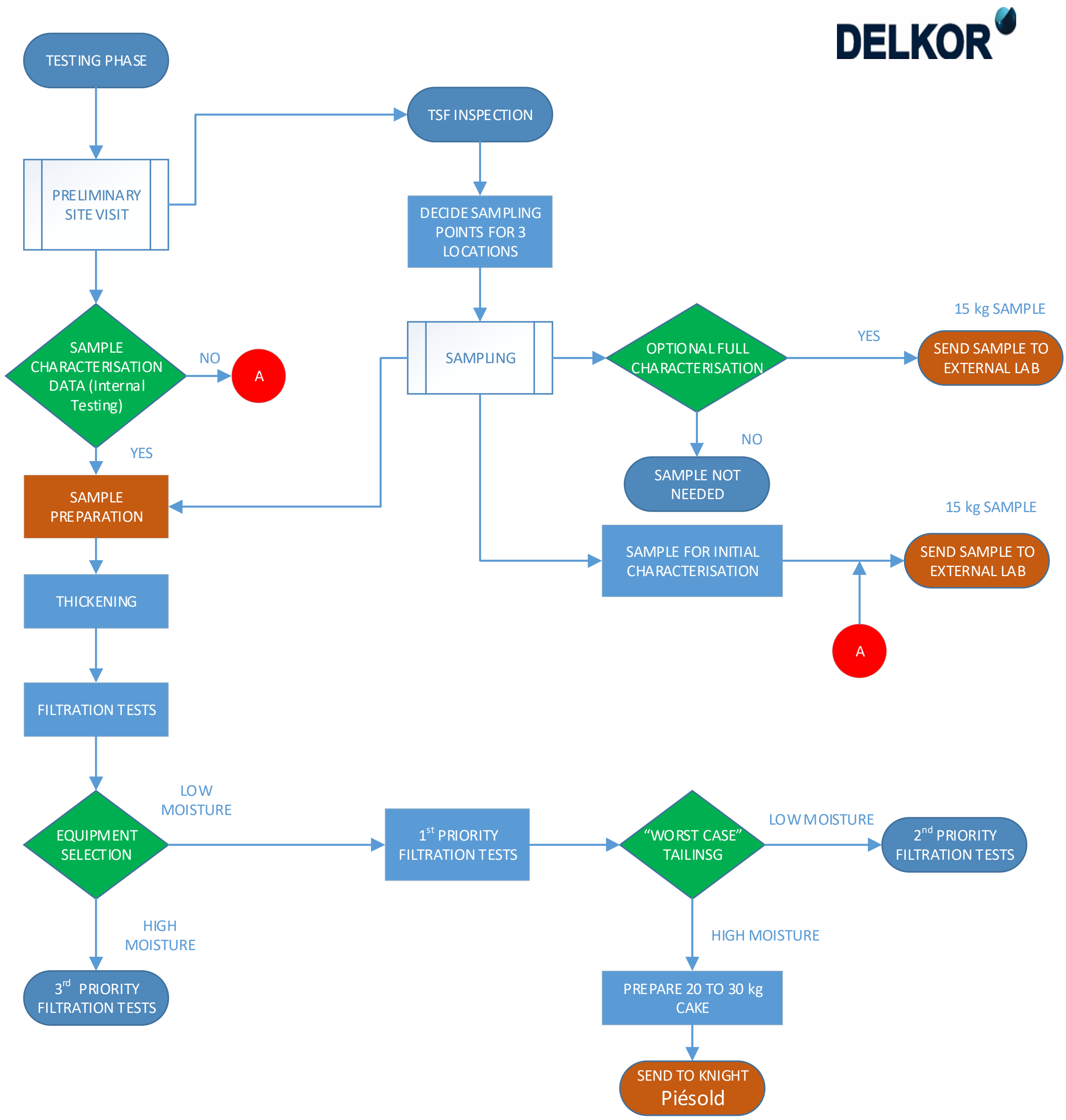


Figure 3 **Solid–liquid separation and sample preparation for geotechnical study**

2.2 Sample characterisation

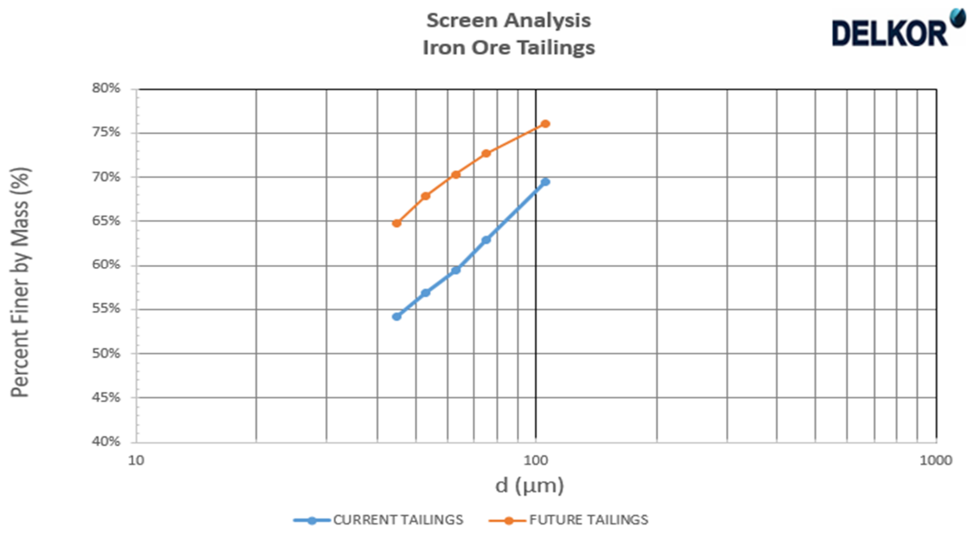
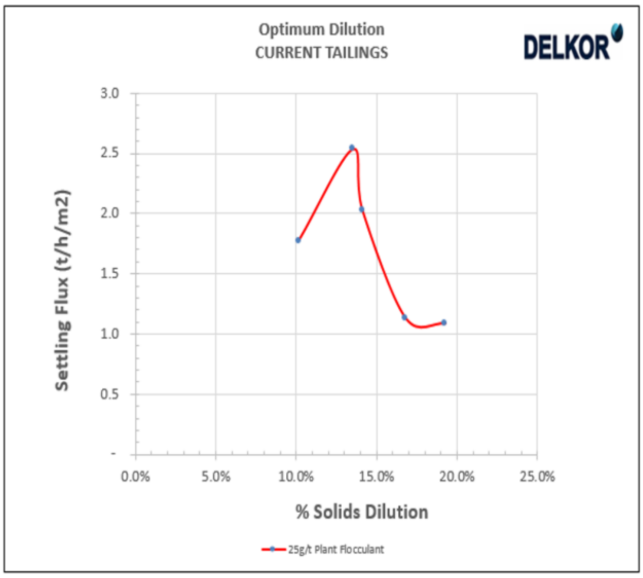


Figure 4 Particle size distribution test results

2.3 Settling and thickening tests

2.3.1 Settling flux

The operating feed dilution is between 10% to 15% solids with thickener flux between 1.7 to 2.3 t/h/m^2. Although the flocculent dosage of 25 g/t is applicable to both tailings materials, additional dosage is required to improve the overflow quality of the future tailings.



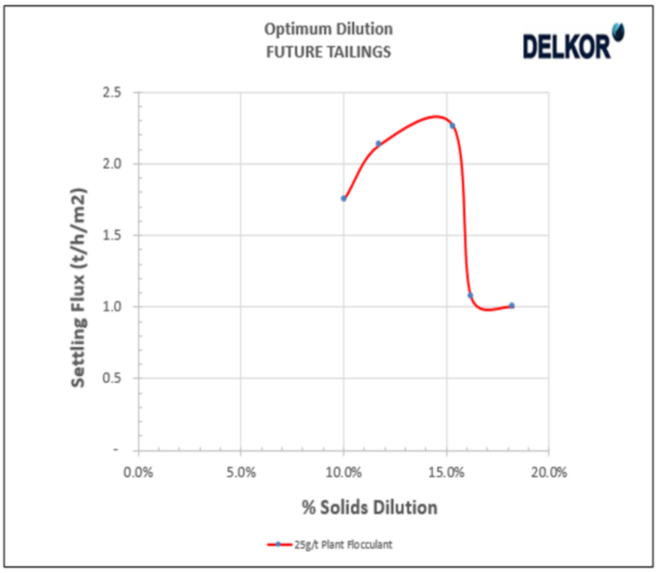


Figure 5 Current and future tailings dilution

2.3.2 Compaction test and rheology

The compaction tests were carried out using a DELKOR paste thickener (190mm diameter × 2,000mm height) at 15 kg/h maximum feed solids capacity. The yield stress of the current tailings underflow slurry containing 73.25 % solids is 101 Pa, while the yield stress of the future tailings is 28 Pa at 68% solids.

Figure 6 DELKOR paste thickener

2.4 Filtration tests

2.4.1 Vacuum filtration

Vacuum filtration tests were carried out at 23-inch mercury (inHg) vacuum pressure using SEFAR TETEX® DLW cloth with air permeability of 400 L/(m^2.s). No further testing was done due to high cake moisture as compared to high pressure filtration.

2.4.2 Pressure filtration

Pressure filtration tests were performed to compare the different parameters (as per Table 1) to produce filter cakes less than 15% moisture.

For future tailings, the filter cake produced with air blow and membrane squeeze was greater than 15 wt % moisture.

A feed density of 55% solids was selected for pressure filtration and final comparison, based on cake thickness (Table 2). The tests were done using 15 bar feed pressure without air blow and membrane squeeze. For geotechnical testing, filter cake samples were generated from future tailings (worst-case tailings with 15.82 wt % moisture as per Table 2).

Table 1 Current tailings pressure filtration parameters

Item	Chamber/feed solids	Feed pressure	Air blow	Membrane squeeze	Cake moisture	Comments
A	50 mm/73%	10 bar	7 bar	12 bar	13.2%	All parameters are within <15% moisture
		15 bar	7 bar	–	13.4%	
		15 bar	–	–	14.0%	
B	50 mm/73%	15 bar	–	–	14.0%	Select 60 mm
	60 mm/73%	15 bar	–	–	14.0%	
C	60 mm/73%	15 bar	–	–	14.0%	The thickener underflow is typically >60% solids
	60 mm/54%	15 bar	–	–	14.8%	
	60 mm/45%	15 bar	–	–	15.3%	

Table 2 Filter cake moistures

Cake thickness	Current tailings	Future tailings
50 mm cake	14.38%	15.77%
60 mm cake	14.69%	15.82%

2.5 Equipment sizing and design consideration

2.5.1 Tailings plant capacity

For this project, our assessment of historical data and future capacities are as follows (refer to Table 3 estimation summary):

- From Figure 7, three standard deviations covered 99% of the production rate in 2019. The red line (top) represents this in Figure 7.
- The real production rate for design consideration is only two standard deviations above the average (see the green line, or second from the top) since many data points are outside two standard deviations below the average (see the purple line). This translates to a factor of 1.27 of the average output.
- For a plant capacity of 1,519 t/h, the maximum value for the life of mine × 1.27 = 1,929 t/h.

Table 3 Tailings plant capacity calculation

Item	Parameters	Data	Comments
1	Average (orange line)	1,383 t/h	Data collected from 1 Jan to 31 Oct 2019 Hourly feed solids (t/h) = feed slurry (m^3/h) and density
2	Standard deviation (STDEV)	184 t/h	Calculated
3	Average + 2x STDEV	1,751 t/h	Green line (selected factor)
4	Tailings production factor	1.27	1,751/1,383
5	Plant capacity	1,519 t/h	Life of mine table
6	Design capacity	1,929 t/h	1,519 × 1.27

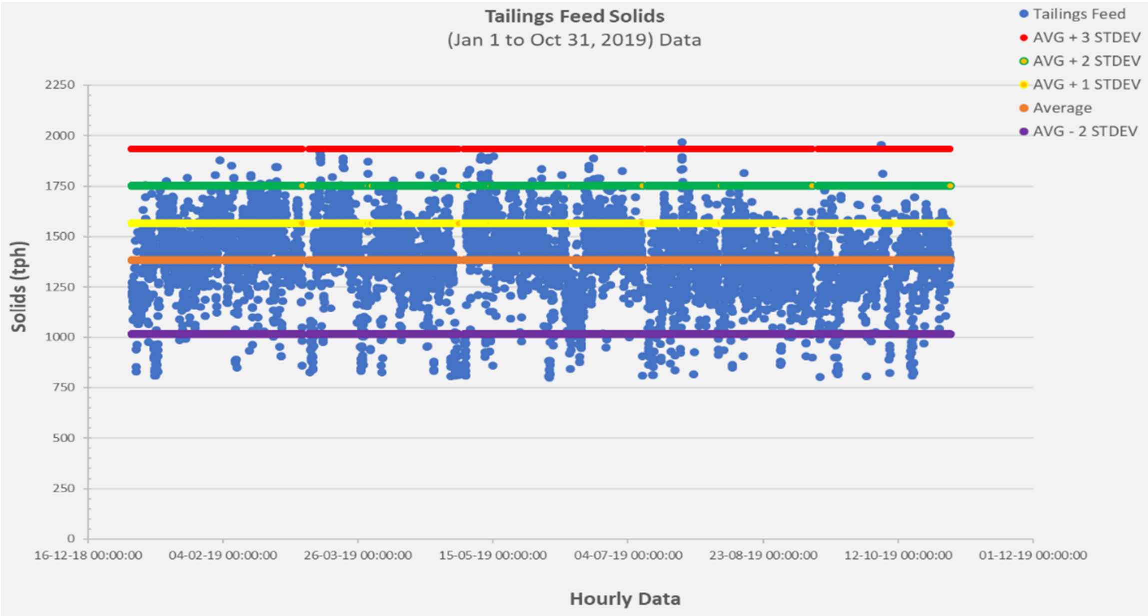


Figure 7 Tailings feed data

2.5.2 Thickener sizing

The operating feed dilution is between 10 to 15% solids with thickener flux between 1.7 to 2.3 t/h/m^2.

- The current two units of 34 m diameter paste thickeners can handle variable feed dilution with its current forced dilution system (Table 4).
- The drive installed can handle a yield stress >150 Pa; the yield stress of the current tailings slurry containing 73.25% solids is 101 Pa, while the yield stress of the future tailings is 28 Pa at 68% solids.

Table 4 Paste thickener design parameters

Parameters	Details	
Existing paste thickener (DELKOR)	Two units of 34m diameter thickeners	
Thickener solids loading	1.0 t/hr.m^2	
Feed dilution	12% forced dilution system	
Thickener underflow	68% solids	
Drive maximum operating torque	5.0 MNm	
Design yield stress capacity	>150 Pa	
Tailings feed	**Current**	**Future**
Feed solids per unit (t)	909	964.5
Thickener loading	1.0 t/h.m^2	1.06 t/h.m^2
Settling flux (test data)	2.3 t/h.m^2	2.2 t/h.m^2
Feed dilution	12% forced dilution system	
Flocculant	25 g/t DRYFLOC 34F	
Thickener underflow (achievable)	73.25% solids	67.96% solids
Underflow yield stress	101 Pa	28 Pa

2.5.3 Filter press sizing

The operating parameter that was selected was 15 bar feed pressure without air blowing and membrane squeeze. Refer to Table 5 for filter press equipment sizing with different feed densities.

Table 5 Filter press equipment selection (60 mm – 15 bar feed pressure)

Current tailings			
Feed solids	73%	55%	45%
No. of units	10	13	14
No. of chambers	185	199	214
Future tailings			
Feed solids	68%	55%	45%
No. of units	11	13	14
No. of chambers	183	205	214

As part of the deliverable on the testing phase for the dewatering tailings, a feed density of 55% solids was considered for future tailings. The initial investment cost of the filter press units excluding the ancillaries is USD 18 million.

- DELKOR filter press model: 14 units of DELKOR FP2000/197/55/15/R/O OH.
- 198 plates/55mm chamber/15 bar feed pressure/recessed type.

2.6 Geotechnical testing

The future tailings sample selected for geotechnical testing was divided and sent to two geotechnical testing laboratories, namely Lab 1 – Golder and Lab 2 – Tetra Tech, where the following tests were undertaken as per Table 6.

Table 6 Laboratory testing schedule

Test	Laboratory	
	Lab 1	**Lab 2**
Atterberg limits	Yes	Yes
Particle size distribution	Yes	Yes
Specific gravity	Yes	No
Standard proctor	Yes	No
Multi-stage consolidated drained triaxial	Yes	Yes
Multi-stage consolidated undrained triaxial	Yes	Yes
One-dimensional consolidation	Yes	Yes
Direct shear	Yes	No
Cyclic direct simple shear	Yes	Yes
Static direct simple shear	Yes	Yes

3 Assessment of tailings conveying and stacking system

3.1 Site visit

The site visit was mandatory to:

- Understand the operation and discuss basic design criteria from the beginning.
- See the future locations of the dewatering plants and tailings storage areas, all in a mountainous area.
- Incorporate aspects of floodwater management, existing infrastructure (e.g. overhead power lines) and geotechnical requirements.

Figure 8 View of the preferred DST area

3.2 Basic design criteria

The basic design criteria included:

- Local data (climate, altitude, temperature range, precipitation, seismic zone).
- Material properties (density after filter press, moisture, swell factor on the conveyor, compaction at the DST, specific ground resistance, DST slope angles etc.)
- Operating parameters (average/maximum tailings feed to filter press, time usage model, nominal conveying capacity, conveyor design capacity, annual tailings capacity).
- Technical specifications for equipment (standards etc.).
- Working regime (hours per shift, shifts per day, working days per week, pit closed days etc.).

Some of the main design criteria for layouting the system are shown in table 7.

Table 7 Basic data (selection)

Data	Unit		Source
Total amount of filtered tailings to be accommodated	Mt	68	Project
Operating life of DST facility	y	9	Project
Density of filtered tailings on conveyor	t/m³	1.9	TAKRAF
Belt speed (max.)	m/s	3.0	Project
Design capacity on conveyors	t/hr	2,284	TAKRAF

3.3 DST pile layout options

The main contractor, Knight Piésold (KP), and the customer developed five options (see table 8) for tailings areas in the preferred topographical boundaries, considering:

- Occupied area.
- Maximum pile height.
- Floodwater and erosion management.
- Infrastructure restrictions (e.g. overhead powerlines).
- Geotechnical safety (10 m single lift height, 30° single slope angle, 10 m lift setback).

(Knight Piésold 2020)

In the next step, TAKRAF included the feasible conveying and stacking alternatives and the proposed DST body suggestions in a pros and cons/trade-off investigation to determine a favourite DST option, taking into consideration (see also table 8):

- Road (max grade gradient of 10:1) and conveyor access to all single lifts.
- Stacking the entire DST area of each single lift with the fewest auxiliary operations (dozing, trucking, compaction) as possible.
- Minimising the conveying lift of material and the final elevation of the DST body to minimise installed power and power consumption of the conveyor drives (OPE).
- A small-sized DST body for the application of a continuous operating system with fewer equipment relocations and modifications (extension, shortening).
- Maintenance of a second dumping area as a backup system as close as possible to the dewatering plant to reduce the number of equipment parts required and the probability of unforeseen stoppages.

Table 8 DST body options and weighting

	Option 1	Option 2	Option 3	Option 4	Option 5
Source	KP	KP	KP	KP	KP/TAKRAF/Client
Plan view (slopes)					
Access to DST area	+++	+++	++	++	+++
Lift of material to…	+668	+678	+750	+654	+668
Compactness of DST body	++	+++	+++	+	++
Backup DST	++	+	+	+	++

Option 5 was the preferred DST.

3.4 Conveying options

The following conveying options were considered:

1. Conventional trough belt conveyors.
2. Pipe conveyors.
3. Ropecon® conveyors.

A pros and cons discussion led to the selection of conventional trough belt conveyors because they enable:

- Unification of the conveying equipment for the entire DST system (alternatives are difficult or impossible to apply directly on the DST pile at the single lifts).
- The possibility of simple conveyor modification (extension, shortening) and relocation, whereas alternatives require special technical and civil construction effort.
- Simple cleaning of the belt required for wet cake and probably sticky filtered tailings material.

3.5 Stacking options

The following stacking alternatives are available:

- A: shiftable belt conveyor + (compact) spreader.
- B: shiftable belt conveyor + single boom spreader.
- C: extendable and relocatable belt conveyor + mobile stacking bridge (MSB) + single boom spreader.
- D: extendable and relocatable belt conveyor + mobile conveyor bridge(s) (MCB) + (compact) spreader.
- E: relocatable conveyor + mobile conveyors (MCs/grasshopper) + radial stacker or spreader.

Alternatives A and B were rejected because:

- Conveyor arrangement and the conveyor shifting scheme are unsuitable for irregular dumping areas.
- Small dumping block volume requires a high frequency of conveyor shifting procedures, generating relative long downtimes.

Alternative C was rejected because:

- Conveyor and MSB arrangement and related dumping schemes are unsuitable for irregular dumping areas.
- A small setback distance from the MSB to the dump slope crest (less than 15 m) is needed, otherwise an additional single boom spreader is required to increase the setback.
- Relocation of the MSB to the next lift is a technical and operational challenge with relatively long downtimes.

Alternative E) was rejected because:

- A high number of single equipment parts increases:
  - The probability of unforeseen technical downtimes (less technical system availability).
  - The number of transfer points, resulting in greater spillage and necessitating increased cleaning and power requirements due to the additional lift of material.
  - The investment costs (estimated as a 1,350 m single conveyor (USD 5.13 million) being replaced by 9 × 150m MCs (USD 9 × 7.7 million).

Consequently, alternative D was selected as the preferred system.

3.6 Time usage model

The application of a time usage model for the entire system and for the equipment parts themselves calculated the Effective Operating Time (EOT). The EOT is mandatory for the estimation of the achievable annual capacity (million t/y), taking into account the nominal tailings feed capacity from the dewatering plant.

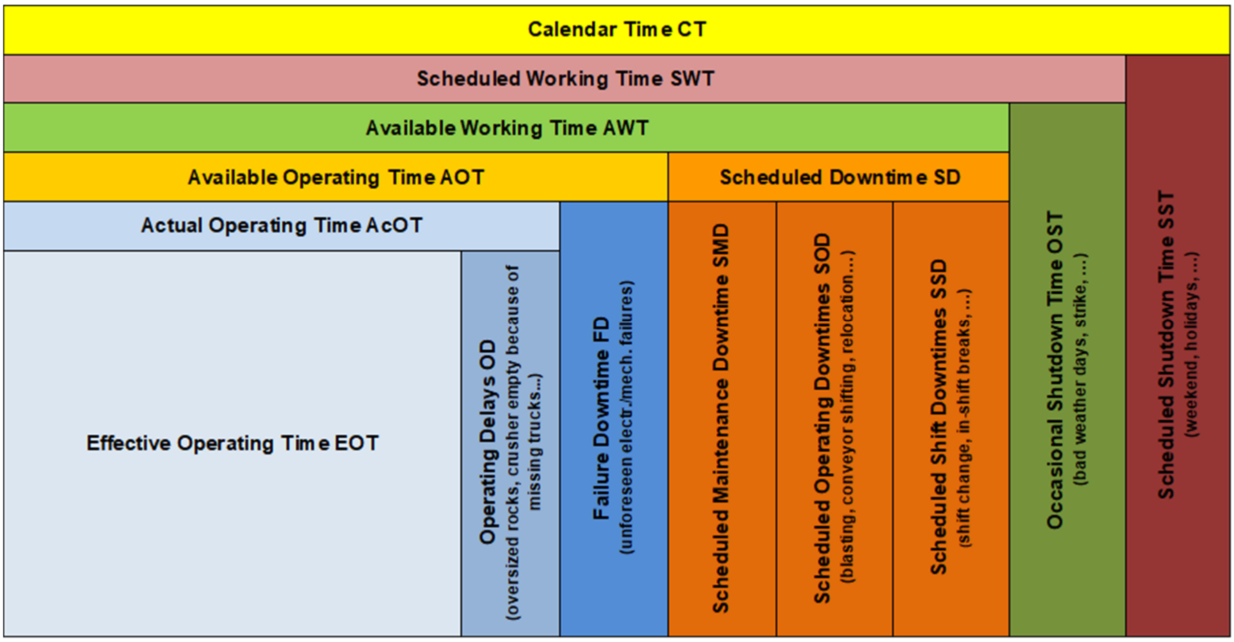


Figure 9 Time usage model

Furthermore, the application of the time usage model for the selected conveying and stacking systems defined the required size of the backup systems.

The dewatering plant, as part of the material processing, delivers a more or less continuous tailings feed to the conveying and stacking system, and stops only for scheduled maintenance and unforeseen downtime of the plant. In this case just 935 h/y overall downtime was calculated, compared to 7,825 h/y of required EOT for the tailings conveying and stacking system.

Performing a multi-lift dump with thin lifts (10 m) requires downtime of equipment for relocation and modification in addition to scheduled maintenance time. As this time is more than the overall downtime of the tailings feeding system, the backup system must make up the difference. The final time usage model for both the main and backup systems provides:

- Required dump body volumes for the main, backup and emergency load-out systems.
- EOT for the main, backup and emergency load-out systems.
- Basic data for investment (capex) and opex calculations for the main, backup and emergency load-out systems.

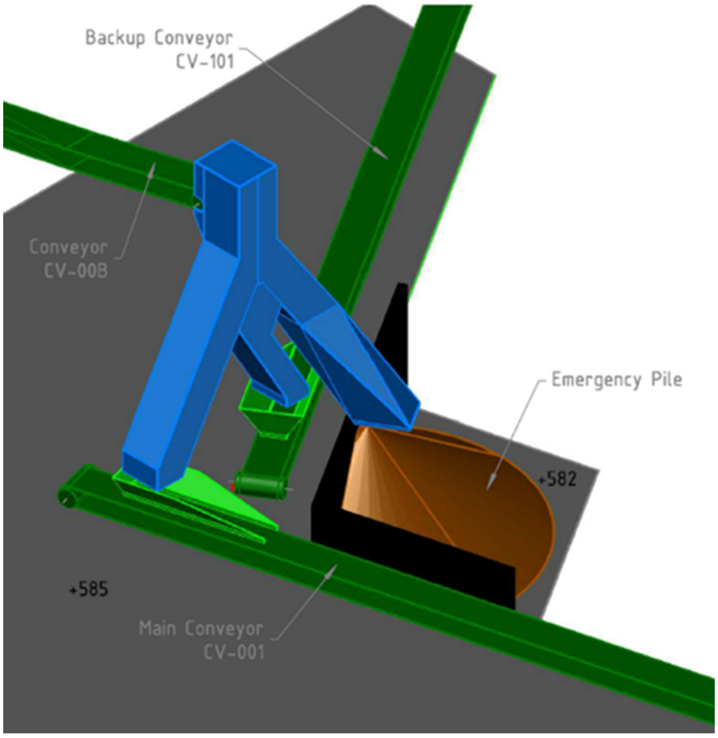


Figure 10 Distribution tower for the main, backup and emergency load-out systems

3.7 Conveying system

Figure 11 shows the layout of the main and backup conveyor systems to the two DSTs.

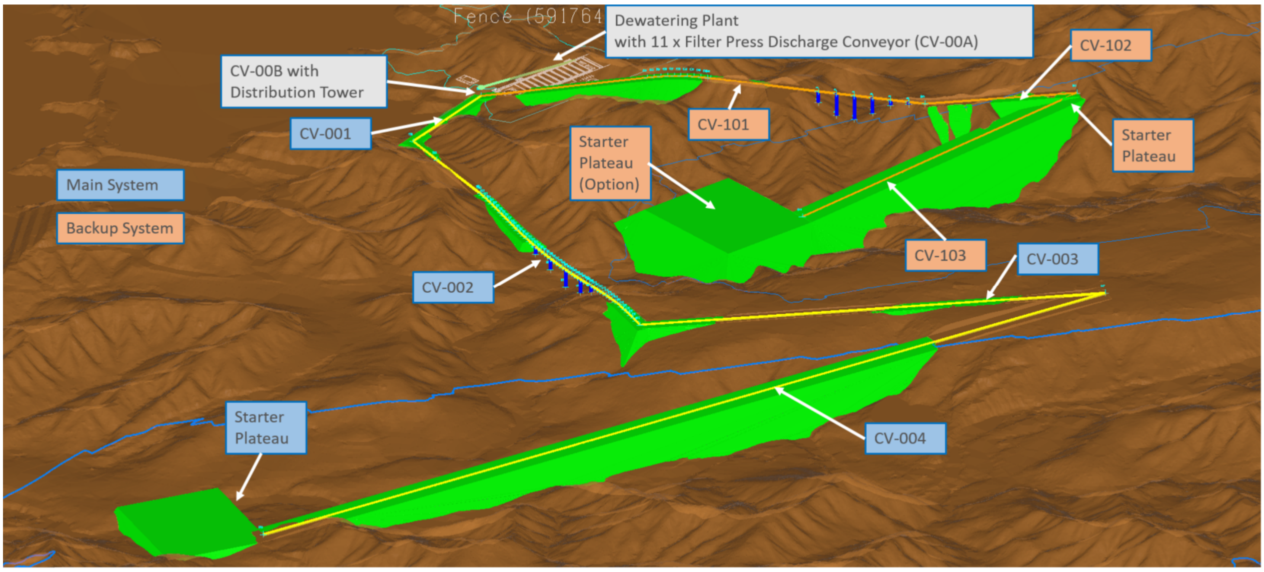


Figure 11 Main and backup tailings conveying systems

3.8 Stacking system

The stacking system can dump 10 m single lifts in an irregular dumping area with relatively low operational downtimes. It comprises:

- One extendable/retractable and relocatable dump conveyor with a crawler-driven mobile head station (CV-004/CV-103 in Figure 11).
- Two crawler-driven MCB, 150 m and 60 m long.
- One crawler-driven spreader.

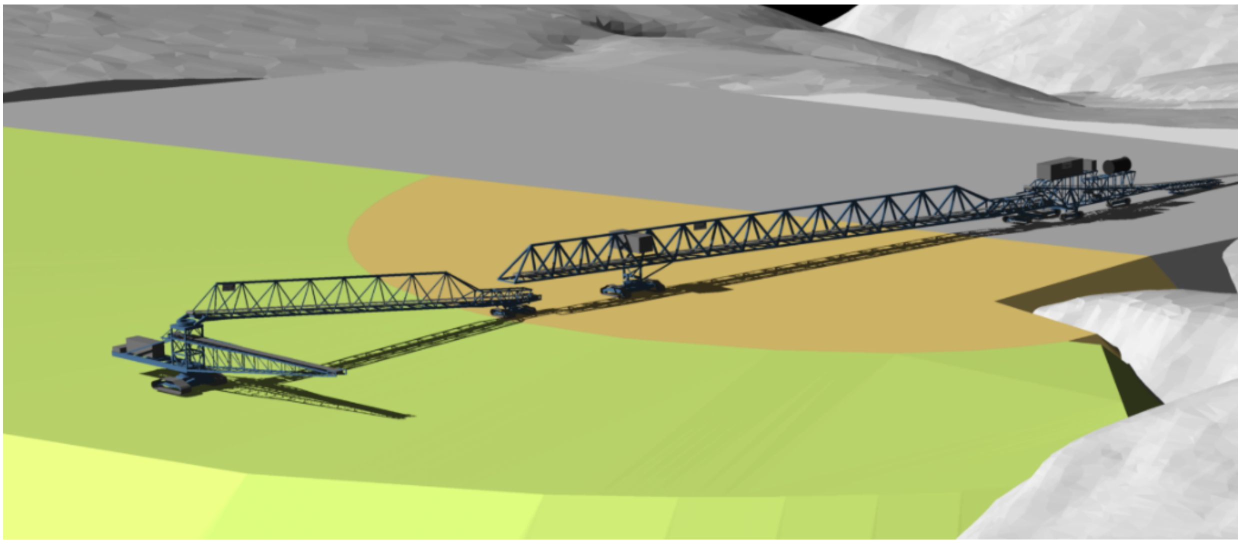


Figure 12 Mobile tailings stacking system

The stacking operation starts with the first lift down in the valley. With every new lift the dump conveyor must be modified according to topography of the DST area to reach the entire area with the mobile stacking equipment, as shown in the dump lift sequences in Figure 13.

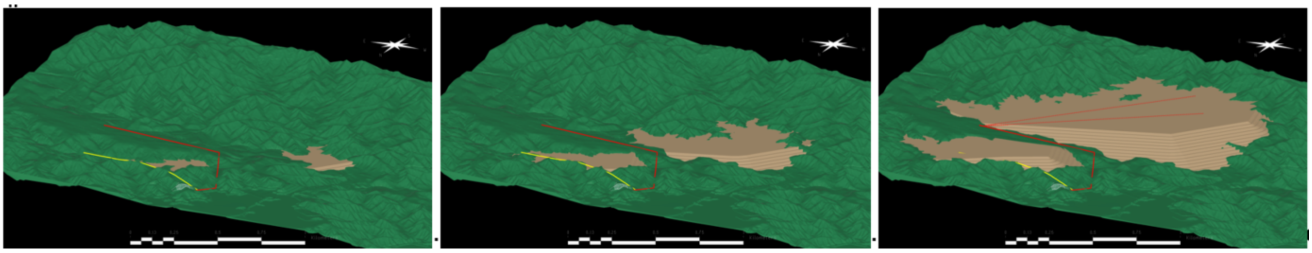


Figure 13 Tailings dump – lifts 3, 6 and 13

3.9 Investment cost

The aim of this case study was the development of a tailings conveying and stacking system for an overall lifetime of nine years, taking into consideration as few as possible major downtimes interrupting the dewatering process. Therefore, all overland conveyors to the designated dumping areas are designed to remain untouched for this time: only the dump conveyor has to be modified, according to the dumping scenario.

This strategy generates an initial equipment investment cost estimated in 2020 at approximately USD 80 million.

3.10 Operating costs

Operating costs were estimated for:

- Equipment power consumption
- Labour for operation and maintenance
- Repair, wear and spare parts
- Lubricants and other consumables
- Auxiliary operations and contingency

The range of opex as listed above was estimated to be between USD 0.76 and 0.80 per tonne of conveyed and stacked tailings. A breakdown is shown in Figure 14.

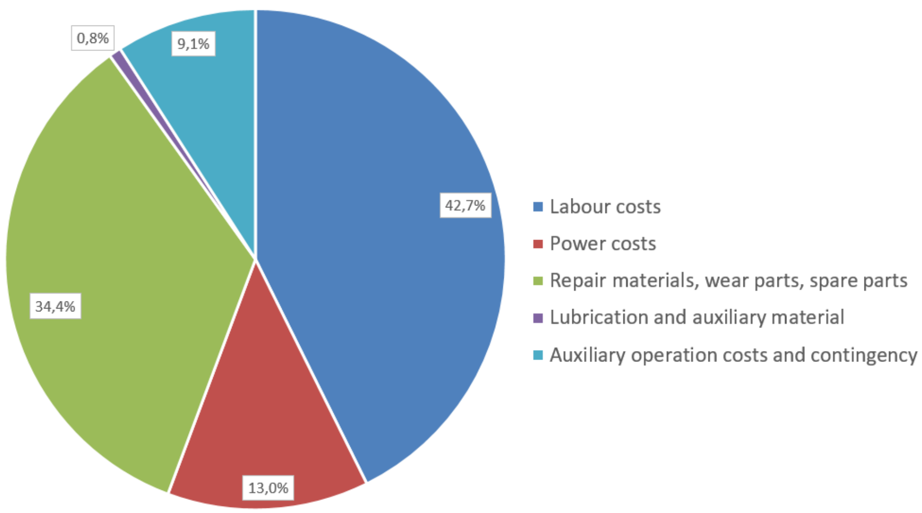


Figure 14 Diagram of OPEX breakdown (here for the main conveying and stacking system)

4 Summary: the pros and cons of the proposed DST solution

The case study investigated a tailings conveying and stacking operation with both a main and a backup system. An additional emergency load-out system caters for upset conditions in the filtration plant, and for occasions when the main system is undergoing maintenance or experiencing operational downtime at the same time as the backup system.

Any solutions with intermediate stockpiles are rejected because of an expected stickiness of tailings with water contents of more than 10% but less than 20%. Stockpile reclaiming or bottom load-out facilities are always critical in handling sticky material.

The conveying systems are designed and calculated for the lifetime of a 130 m-high 13-lift DST.

The stacking system was selected to enable dumps in 10 m lifts in irregular dumping areas with a safe front slope and natural topography behind, and with as few as possible transfer points. The mobile dumping equipment itself allows a large setback distance to working slopes and high mobility in the event of emergency travel due to possible slope failures (weather events, earthquakes).

The advantages against any other equipment solutions, and the installation of smaller conveying and stacking systems, initially comes with higher extension costs for future lifts.

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