Chair of Mining Engineering and Mineral Economics

Master's Thesis

Comparison of Mining Transportation Systems from Environmental Point of View

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Abstract

High demand for mineral resources, meeting daily production targets, and low diesel prices have marginalized energy management in the mining industry for many decades. Nevertheless, rising energy costs and recent legislative and societal demands to reduce greenhouse gas (GHG) emissions are forcing mining companies to improve energy efficiency. These incentives are gradually becoming more influential determinants in the choice of methods and equipment types for the various mining operating practices. This study aims to compare environmental impacts of continuous with conventional mining, including dust, noise and vibration with a special focus on energy consumption and lifecycle GHG emissions of different truck and belt conveyor configurations. The thesis developed a calculation tool for truck energy consumption and GHG emissions, which was validated by measurement data from a surface coal mine in Queensland, Australia. German industry standard DIN 22101 was used to calculate the energy consumption for belt conveyor systems, and corresponding operational data from the Hambach opencast lignite mine to confirm the results. In order to compare the energy efficiency and lifecycle GHG emissions for in-pit crushing and conveying (IPCC) systems with diesel, electric, and trolley assist truck haulages, different parameters have been investigated including production rate, haulage distance, slope grade and rolling resistance, type of material, source of electricity and fuel type, and capital and operating cost. The results indicate that in most cases, trucks require more energy to transport the same amount of material and consequently cause more severe environmental impacts including GHG emissions, even if they are electrically powered. Moreover, a proper IPCC setting can be more cost-effective, particularly when it comes to large deposit at certain depths.

List of Tables

List of Figures

Table of Contents

1 Introduction

1.1 Overview

The definition of sustainable development is profoundly dependent on the context in which it is used. Concerning social, environmental, and economic development, the United Nations defines a set of goals and principles to support meeting the present needs without undermining the integrity and stability of the natural system in the future. The origin of sustainable development lies in ecological issues, such as forestry (Assembly, 2015). A typical forestry application referred to the number of trees, which simply guaranteed the size of forests by declaring that the number of trees harvested must be equal to or less than the number of trees farmed (Grober, 2007).

In most cases, the approach is not as simple as setting a ceiling on the harvest of trees in a forest. When it comes to the earth's non-renewable resources, the idea of depletion fundamentally contradicts the concept of sustainability. To be more specific, in a mining operation, the development involves taking material from the earth and transporting it to another destination, sometimes a continent away. Therefore, it is crucial to set realistic goals to achieve successful interaction with the ecosystem. As long as the recovery of minerals already extracted and used cannot satisfy the increasing demand for raw materials, the difference has to be made up by mining the earth's resources. This explains why, in the context of mineral resources and other non-renewable resources, the concept of sustainable development is more stand and from on the idea of development rather than sustainability. Accordingly, it focuses on the operation itself, in this case, mining, instead of dealing with the recovery process of the resource. This means by accepting the fact that the mining operation is inevitable, the concept strives to establish principles to reduce the environmental impacts. In the example of forestry, on the other hand, sustainability is more tangible. In this case, the concept of sustainable development concerns both the environmental impact of timber harvesting and the process of recovering trees as renewable resources of the earth. Accordingly, sustainable development goals for non-renewable earth resources can be defined as reducing the environmental impact and increasing the recovery rate of the resource to defer the need to extract new resources.

In general, environmental impacts of a mining operation include disturbance of the primary state of undeveloped or greenfield land, greenhouse gas emissions, and acid mine drainage, as well as temporary impacts such as dust, noise, and vibration. A modern reclamation plan provides solutions to most environmental impacts and consequences from the earliest days in feasibility studies, through the life of the mine, and after the mining extraction process is complete. Loading and hauling systems are responsible for most GHG emissions in a mine site. The performance of these two systems is essentially subject to production planning and optimization targets, i.e., maximizing the project's net present value, which may explain why the management of energy efficiency and GHG emissions has been overlooked or excluded.

1.2 The goal and objectives

The goal of this work is to compare the environmental impacts, especially greenhouse gas emissions, of different configurations of two main mining transportation systems and their possible combinations. To this end, the following objectives have been set:

- Development of key parameters to compare IPCC and shovel/truck equipment.
- Development of a calculation tool to determine the energy efficiency of both mining methods on a high-level basis.
- Comparison of the environmental impact in terms of emissions, e.g. CO_2, dust, noise and vibration.
- Analysis of potential implementation of renewable energy solutions to reduce the carbon footprint of mining activities.

1.3 Thesis outline

Following an overview of two transporting systems, a life cycle assessment example of crude oil, and environmental legislation background, chapter three briefly introduces the different parts of the crushing and conveying system in surface mining. Afterward presents the dust, noise, and vibration for common operational conditions. Using the German DIN 22101 standard for calculating the energy consumption of conveyor systems and operational data from the three RWE opencast coal mines, some empirical constants were adjusted. The effects of various parameters such as length, grade, material type, and utilization were analyzed and represented.

Chapter four discusses operating parameters for haul trucks. It explains why conventional time-based methods underestimate the fuel consumption of a truck and thus of the entire transport fleet, and energy requirements are calculated using two approaches based on cycle time and the time-independent method based on road specifications. Finally, noise and dust emissions are presented.

Chapter five compares transportation system options using a common unit: kWh per ton and kilometer. For truck transportation systems, the LCA for diesel and biodiesel was used. Since IPCC systems use electricity as their primary energy source, the LCA method was applied to various electricity sources, including lignite, coal, petroleum, natural gas, photovoltaic, biomass, nuclear, hydropower, and wind power. A model was developed that includes the equations and formulas for both systems to facilitate the comparison of energy use and greenhouse gas emissions for potential material handling options. For a project plan to be successfully implemented, it must be economically feasible; otherwise, it is not truly sustainable. A cost model was developed to compare the capital and operating costs of systems for different production rates.

The summary of the study are presented in Chapter six.

2 Literature Review

2.1 Continuous mining system

The merger of the Rhenish lignite mining district in 1959 provided the impetus for the company now known as RWE to develop and design new mining systems and equipment types. Bucket-wheel excavators with a rated capacity of up to 240,000 cubic meters per day, belt conveyors, and spreaders were the main components of the continuous mining system that has been used in open-cast and large-scale mining operations ever since (M. Schmitz, Franken, & Blunck, 2010).

These components remain mechanically connected during loading, conveying, and dumping the material. Human intervention is mainly needed for maintenance and control rather than operating. This is important as it reduces labor costs and increases safety by removing more people from high-risk environments. Accordingly, the system is suitable for automation and can readily be compatible with advanced monitoring and measurement technics and devices. Studies in recent years have shown promising application practices for sensors to collect data from conveyors and evaluate features such as calorific value and ash content in coal mines (Benndorf & Et al, 2015).

In most cases, the material must be crushed before being transported by conveyor belts, which means that a primary crusher must be used in the mine. For loose material, double roll crushers are installed in bucket wheel excavators to crush the material for transfer to the tripper car. For solid material, loading usually occurs after drilling and blasting. Shovels, loaders, and draglines are the most common choice for loading such material, and they can feed the material directly into a crusher or load trucks.

Based on the mobility of the crusher, the conveying methods are divided into three general systems. When the crushers can be moved frequently and follow the loading unit efficiently during the shift, the system is named mobile in-pit crushing and conveying system. When the crusher relocates on the order of years, it is called semi-mobile in-pit crushing and conveying system, with locations determined according to the requirements of medium-term production planning. If the crusher remains at its location for more than ten years and is relocated under the necessities of long-term production planning, the system is fixed in-pit crushing and conveying.

Although the mentioned approach for dividing is not precise, it is rather helpful for equipment selection and mine design. In fact, the location of an in-pit crusher is an operational research question that has been investigated with different methods and techniques. An optimal solution improves the net present value of the project by reducing mining costs. In addition, a compatible combination benefits from truck flexibility and conveyor reliability, reducing financial and operational risks.

The use of in-pit crushing systems in the mine is gaining importance as the need for efficient and practical transportation is more necessary than ever. While the technical aspects have been studied over the last decades, there is still much room for studying the environmental impact of each configuration, especially the CO2 emissions of the systems.

2.2 Conventional mining system

The size of a mining operation defines its required type of equipment. The haulage system, as a case in point, is firmly determined by the mine's annual production. Even the production of a small size open-pit mine could not be attained feasibly by a typical on-highway truck. Different types of off-highway trucks have been used in mining operations to transport materials from loading units to crushers, dumps, and stockpiles. In addition to size, the haulage distance as another equally important factor directly impacts loading and hauling fleet combinations. The comprehensive transporting system determination requires an accurate and precise judgment of different possible choices. Three types of trucks commonly used in surface mining operations are presented below.

2.2.1 Articulated dump truck

An articulated dump truck has three axles with an articulation point between the front axle and the two rear axles (see Figure 2-1). The three axles all provide power to the wheels and could be adjusted according to uneven terrain conditions. The larger contacted area to ground distributes the pressure of the load among the tires. As a result, an articulated dump truck provides improved traction and flotation in poor underfoot conditions. The payload of these trucks generally ranges from 24 to 41 tonnes (Caterpillar, The 48th Caterpillar Performance Handbook., 2018). Altogether, articulated dump trucks are a better option for small mines with poor underfoot and high slope grade terrain or as an auxiliary truck in large scale surface mines where they can cover the inflexibility of bigger vehicles.

Figure 2-1Articulated dump trucks: Volvo A40G 4o t (Volvo, 2021), and CAT 745C 41 t (Caterpilar, 2021)

2.2.2 Bottom-dump hauler

Truck-trailer and unibody are two configurations of Bottom-dump haulers that have modified chassis and trailer (see Figure 2-2). Typically, Truck-trailers range from 90 to 136 metric tones, and their extended chassis limits them to operate up 10 % of effective gradient. To outweigh the extra capital cost of adjustments, it is expected to be used for transport distances over 3km. Unibody type has higher energy efficiency and payload ratio, enabling them to maintain higher speed and shorter cycle time. Compared to corresponding rear dump trucks, the 50-70 % higher payload makes Bottom-dump trucks an appropriate choice for shallow deposits with a relatively low gradient profile, such as coal mines needed to deliver extracted materials to a nearby power plant (Humphrey & Wagner, 2011).

Figure 2-2 Kress 200Clll coal hauler, 220 tonnes (KRESS, 2021).

2.2.3 Off-highway rigid-frame trucks

Rigid frame trucks are the most common type of trucks that have been using in open-pit operations. They have two main drive train systems, mechanical drive and electrical drive; depends on the type of electrical current electrical drive trucks are divided into AC and DC types. Electric drives consist of an engine, generator, power converter, wheel motors, planetary gear sets, and retarding grid .The generator converts the diesel engine's mechanical energy to electricity used by wheel motors afterward. The mechanical set has better driving performance in steeper grades with higher speed, while in retarding mode, some electric drive trucks could generate and save electricity. The capital cost of electrical drives is higher, but the operating costs are lower because some parts are less worn.

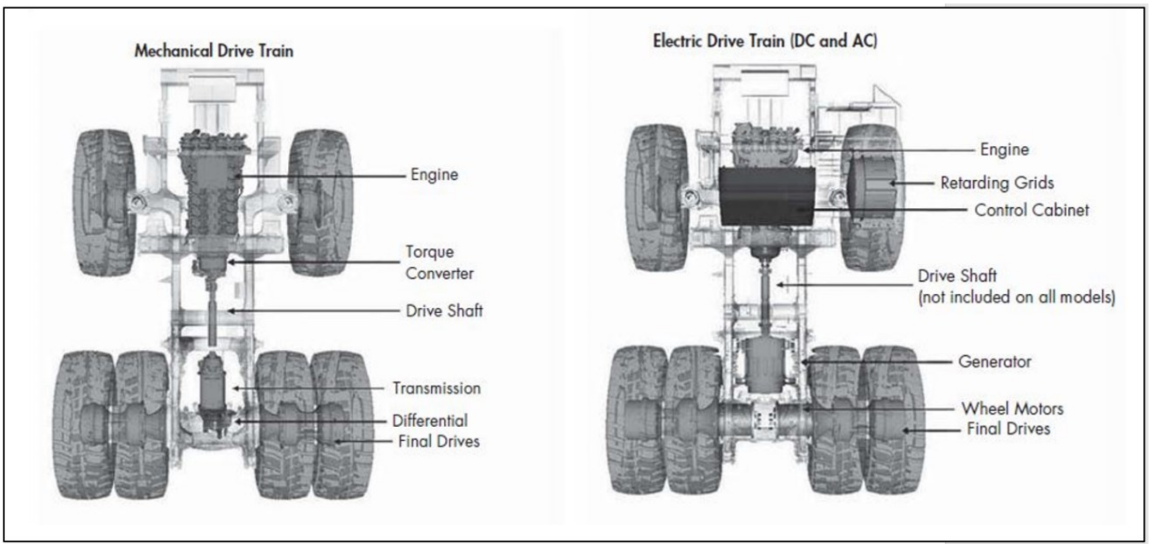


Figure 2-3 Trucks power train systems (Humphrey & Wagner, 2011).

2.2.4 Trolley-assist mining truck

Trolley-assist mining truck is an electric drive rigid frame rear dump truck which draws the electricity from overhead power grids when it moves uphill. Almost 70-80 % of entire cycle fuel consumption, burns while the trucks drive uphill. When the trucks attached to a line the can increase their speed up to 70 %, means the cycle time and number of trucks are reduce.

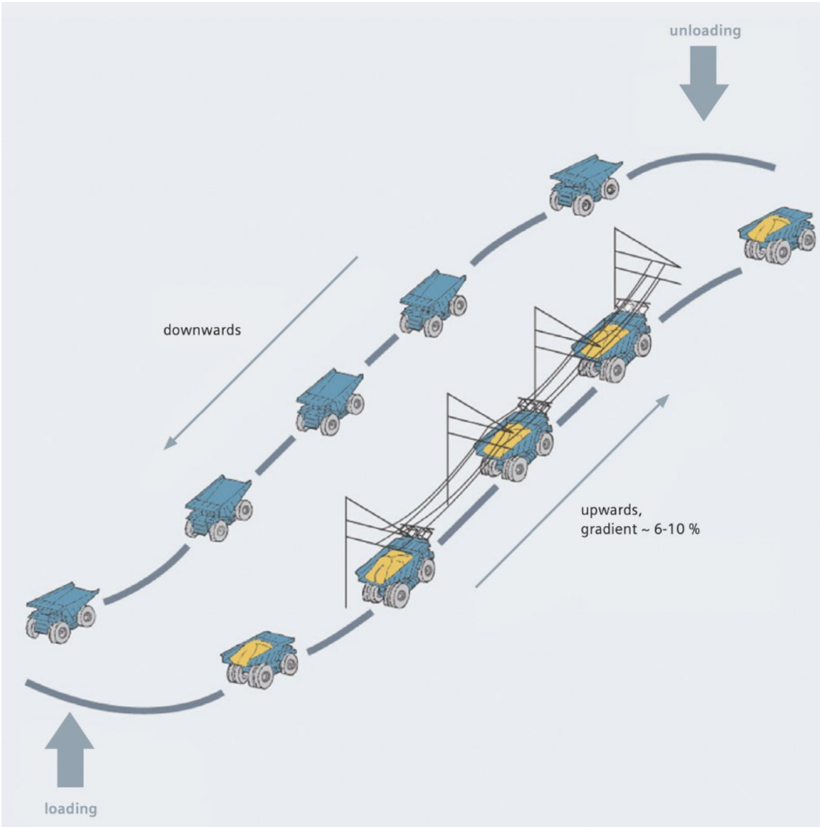


Figure 2-4 Truck trolley system simplified model (Siemens, 2021).

The lifetime of truck increases and the CO2 emission and noise reduce significantly. This is also highly depends on the source of electricity. Since the system of grids are quite inflexible, they should be designed for being fixed for 5 to 10 years, similar to semi mobile IPCC system.

Figure 2-5 Trolley assist trucks (Caterpillar, 2020).

2.3 Greenhouse gas emissions

According to the Kyoto protocol, greenhouse gases constitute seven gases that contribute to global warming. They are carbon dioxide (CO2), methane (CH4), nitrous oxide (N2O), and four fluorinated gases: hydrofluorocarbons (HFCs), per fluorocarbons (PFCs), sulfur hexafluoride (SF6), nitrogen trifluoride (NF3) (Glassary: Kyoto Protocol, 2022).These gases have different Global Warming Potential. In other words, the heat amount they can absorb is not the same. To be able to calculate and analyse their individual and total contributions to global warming in the next hundred years, the CO2 equivalent can be used as a metric measure (Becken, 2022).

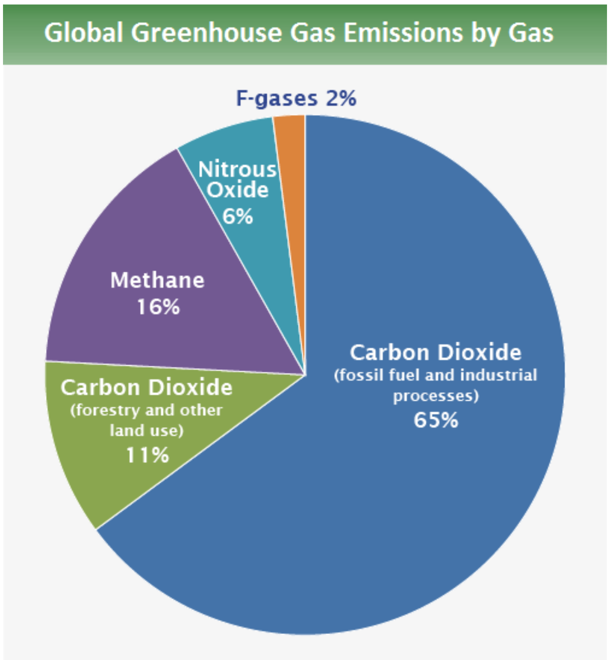


Figure 2-6 Global share of greenhouse gas emissions (IPCC, 2014).

Table 2-1 GHGs' atmospheric concentration, lifetime and GWP (CARO, 2022)

GHG	Atmospheric Concentration	Atmospheric Lifetime	Global Warming Potential (Over a 100 year period)
Carbon Dioxide (CO2)	76 %	1000's of years	1
Methane (CH4)	16 %	10 years	25
Nitrous Oxide (NO2)	6 %	> 100 years	298
Fluorinated Gases	2 %	1,000 - 10,000 years	1,000 - 10,000

2.4 Life cycle assessment and GHG emissions

Life cycle assessment is a method for evaluating the cumulative environmental impacts of a product, process, or service across the industrial value chain. Defining cycle boundaries is an important and challenging task that genuinely impacts the outcome. This study uses LCA data for comparing the different energy sources' GHG emissions. For fossil fuels, therefore, the assessment takes into account GHG emissions from on-site consumption as well as transportation, extraction, and other facilities-related activities.

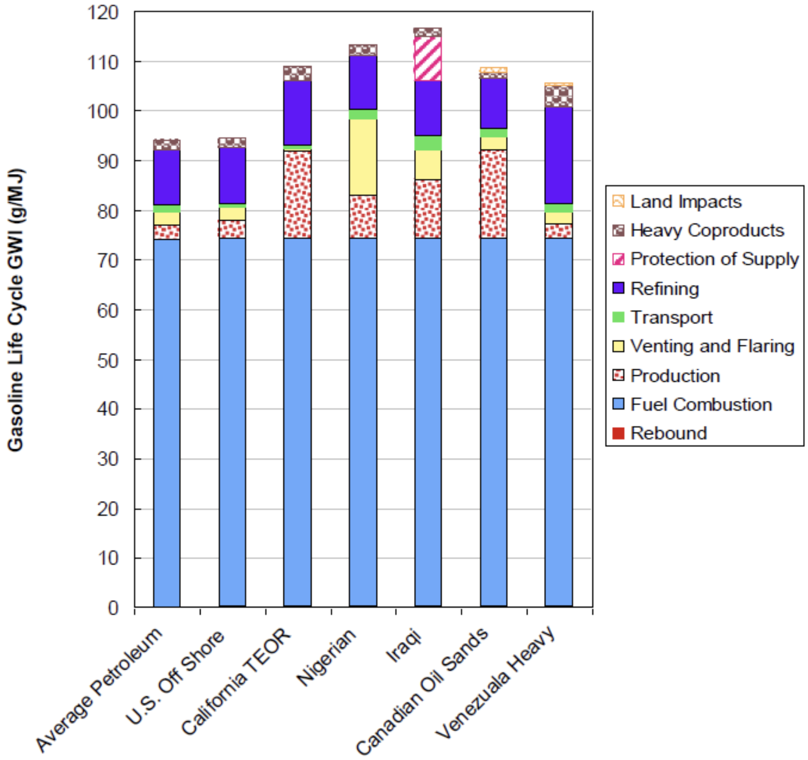


Figure 2-7 Summary of GHG Emissions for Different Crude Oil Production Scenarios (Unnasch, Wiesenberg, & Sanchez, 2009)

Table 2-2 Lifecycle GHG Emissions of Various Electricity Generation Sources (World Nuclear Association, 2010).

Technology	Mean	Low	High
	tonnes CO2e/GWh		
Lignite	1,054	790	1,372
Coal	888	756	1,310
Oil	733	547	935
Natural Gas	499	362	891
Solar PV	85	13	731
Biomass	45	10	101
Nuclear	29	2	130
Hydroelectric	26	2	237
Wind	26	6	124

2.5 Energy intensity in mining industry

In addition to the energy source, the amount of energy consumed is an influential factor contributing to environmental impacts, particularly GHG emissions. Traditionally, energy management has been overlooked in the mining industry, mainly because of the importance of production plan targets and low energy costs. On the other hand, the ore bodies are becoming more challenging to access as the pits get deeper and more energy is needed to develop deposits and transport materials. In addition, raw material prices are rising, and advances in processing technology are making it possible to mine ores with lower cut-off grades, requiring more energy to process.

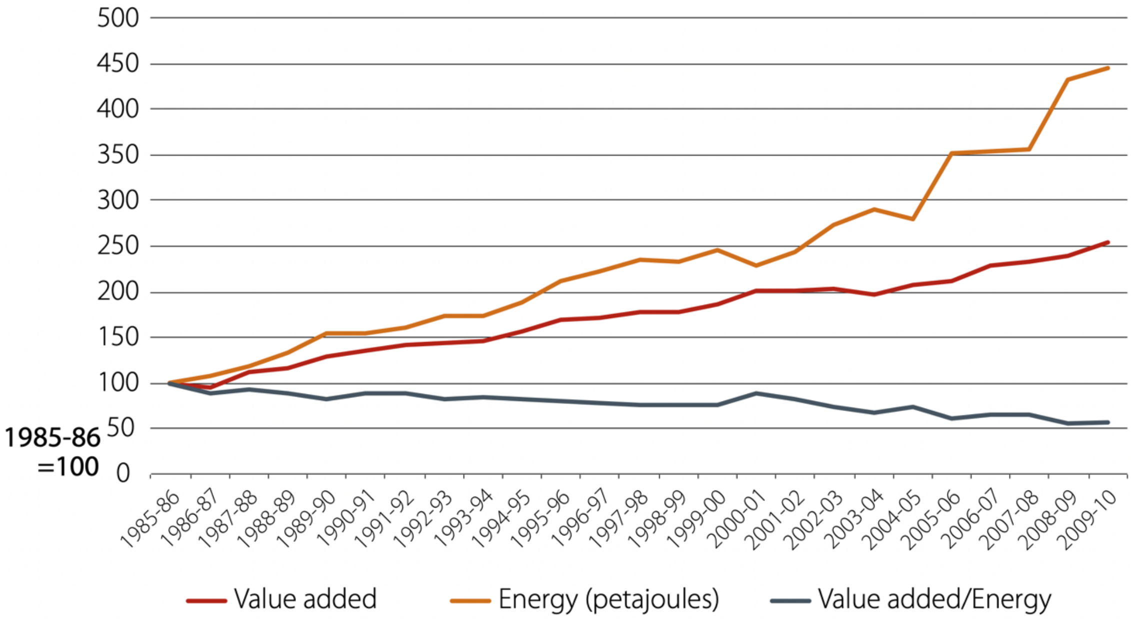


Figure 2-8 Trend of energy productivity in the mining industry (A Syed, 2013).

2.6 Reclamation

In the United States in the late 1930s and early 1940s, some states began to enact laws to regulate coal mining, which had begun in 1740. The high demand for coal during the war eased these concerns and led to continued coal mining, neglecting environmental impacts. Over time, mining companies have been required to obtain permits and post bonds to ensure that future reclamation costs are covered. However, differences in state regulations allowed mining companies to relocate to states with less stringent regulations. In August 1977, the law was finally signed as one of the President-elect's campaign promises. Under the Surface Mining Control and Reclamation Act of 1977, mine reclamation is the process of converting mined land to an environmentally beneficial or economically usable state. Accordingly, the consequences are described as: “Many surface mining operations result in disturbances of surface areas that burden and adversely affect commerce and the public welfare by destroying or diminishing the utility of land for commercial, industrial, residential, recreational, agricultural, and forestry purposes, by causing erosion and landslides, by contributing to floods, by polluting the water, by destroying fish and wildlife habitats, by impairing natural beauty, by damaging the property of citizens, by creating hazards dangerous to life and property by degrading the quality of life in local communities, and by counteracting governmental programs and effects to conserve soil, water, and other natural resources" (30 U.S.C. 1201 & following).

In Germany, the first legal basis was established in the 18th century, mainly concerning the reclamation of excavated land and covering it with topsoil for agricultural purposes. As the demand for coal and other minerals increased, more scientific research was conducted, and more reclamation methods were developed. By the end of the 20th century, the first state agency was established to rehabilitate nearly 80 abandoned mines and greenfields. On 13 August 1980, the federal legislature transferred the incoherent number of state regulations on mining into a uniform regulatory law (Umweltbundesamt – UBA, 2021). The Federal Mining Act (BBergG) defines three primary purposes:

"1. to ensure the availability of raw materials by managing and promoting the exploration, extraction, and processing of mineral resources with a view to geographical constraints and sustainable mining while applying economic and low-impact technology,

2. to ensure the safety of mining operations and employees, and

3. to strengthen precautions against risks to human life, health or to third-party equipment and materials arising from mining activities and to improve the compensation of unavoidable damage".

While Section 2, Material and territorial scope of application, clarifies that the act applies to methods of transportation in mines as long as they are directly related to exploration, extraction, or processing, the same section excludes almost all modes of transportation, including rail systems, automobile transportation, maritime traffic, aircraft, and pipelines, that are not intended for the exploration, extraction, or processing of freely mineable and proprietary mineral resources (BGBl. I S. 1310).

3 Environmental assessment of the conveying systems

3.1 Introduction

Today's mining companies have to extract minerals from low-grade deposits located at depth and difficult to access with higher production per year. Having a higher stripping ratio and longer operation lifetime compared to the past, requires innovative and cost-effective approaches that could also tolerate price fluctuations. For the past few decades, the question has been the transition depth to change an open-pit mine to an underground, nowadays, the question changed to the transition depth to convert a shovel truck system to an IPCC one. On the other hand, depending on the energy source, IPCC systems have the opportunity to reduce their environmental impact significantly, this becomes more important as future mining permit requirements take current environmental concerns into account. Belt conveyors are an essential part of continuous and IPCC systems; the shows the components of a typical belt conveyor.

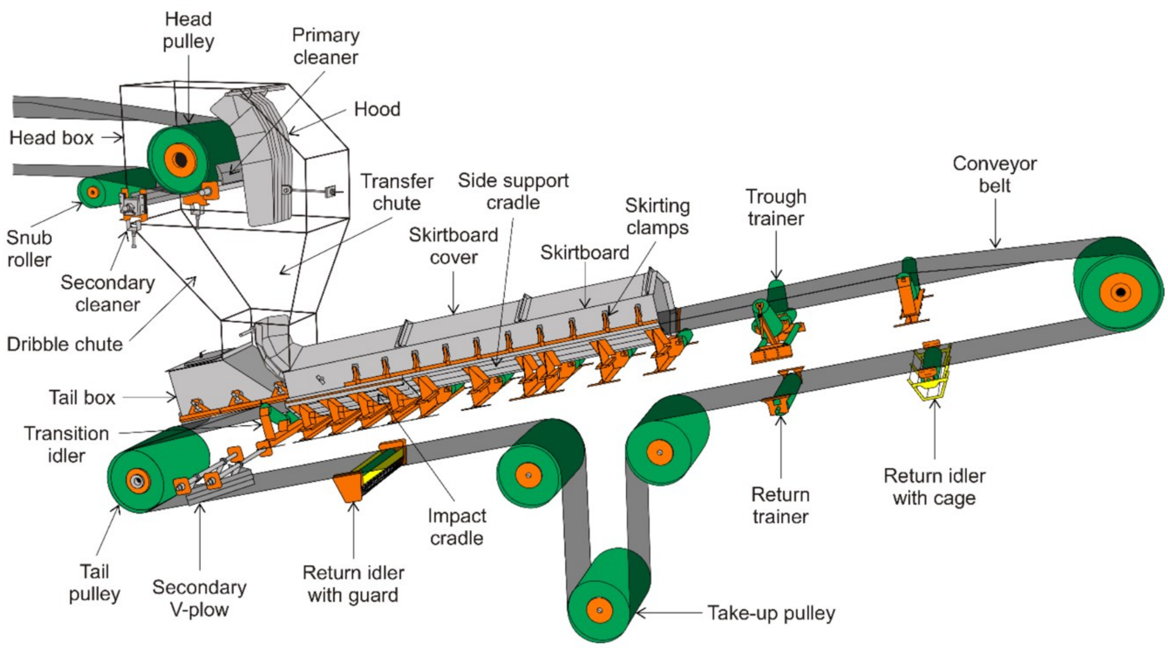
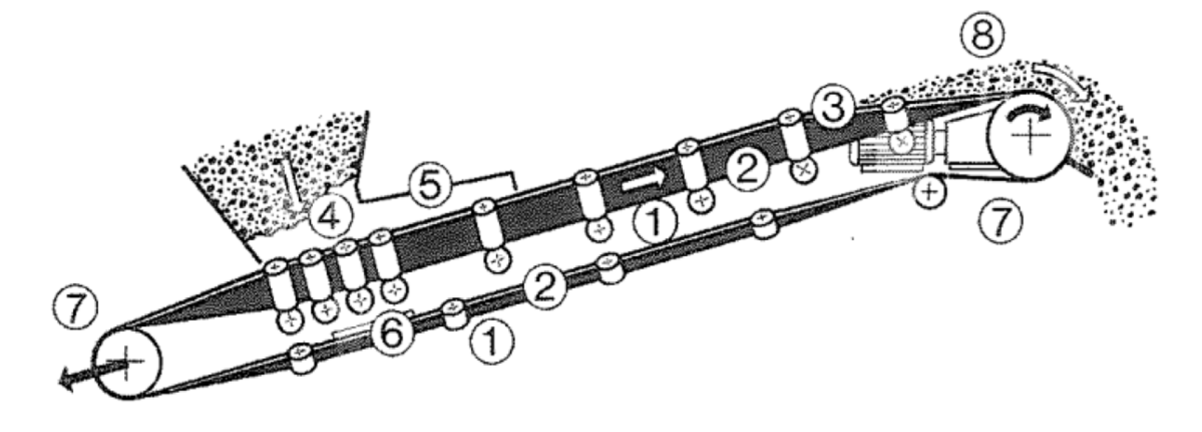


Figure 3-1 Belt conveyor´s components (Cecala & al, 2019) Illustration adapted from ASGCO.

3.2 Belt conveyors energy calculation

The first step for calculating the required energy for belt conveyors is to estimate the minimum forces to overcome all the other resistance forces. Figure 3-2 presents eight forces that have to be overcome by the traction force as an electrical motor’s output.



1. Rolling resistance of idlers
2. Belt flexing resistance
3. Flexing resistance of material conveyed
4. Feeding resistance
5. Skirting friction resistance
6. Scraper friction resistance
7. Belt deflection resistance at pulleys
8. Gradient resistance

Figure 3-2 Resistance forces for a belt conveyor (Rainer, 2008).

These forces can be simplified and divided into four main categories: F_H, F_N, F_{ST}, and F_s, see the equation 3-1 below.

$$F = F_H + F_N + F_{St} + F_S \quad \text{Equation 3-1}$$

F is the minimum needed force.

F_H is the primary force corresponding to the mass of load and the frictional forces for upward and return runs. According to DIN standard 22101:

$$F_H = f\,.\,l\,.\left[\, m_R^l + \left(2\ m_G^l\ +\ m_L^l\,\right).\ cos\,\delta\,\right].\ g \quad \text{Equation 3-2}$$

f: resistance coefficient

l: length of conveyor

m^l_R: is the mass of idlers, which corresponds to the force required to move the idlers in top and the return runs, and is reasonably related to the number of idlers and their rolling resistance.

M^l_G: is the mass of the belt itself, including the cover and the tension member inside. M^l_L: is the mass of material.

g: gravitational acceleration.

F_N: is the secondary resistance force, results mainly from frictional and acceleration forces in the feeding zone. Using the C coefficient, F_N can be calculated as a ratio of the F_H. The amount of C depends on the length of the conveyor as figure xx indicates.

$$F_N = (C - 1).F_H \quad \text{Equation 3-3}$$

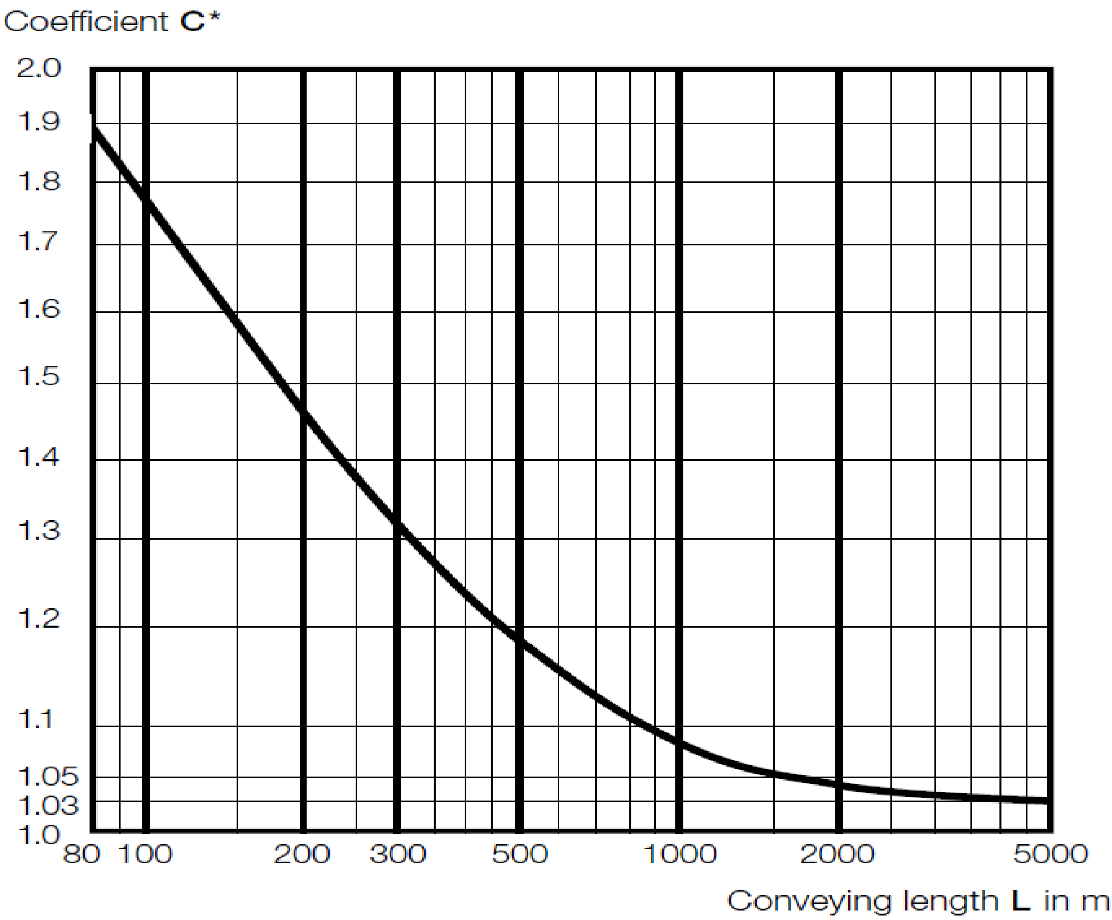


Figure 3-3 Conveying length and C coefficient (Rainer, 2008).

F_{St}: is the gradient resistance force:

$$F_{St} = H.\ m_L^l.g \quad \text{Equation 3-4}$$

Finally, the F is the minimum required force, as mentioned above the 3-1 is a simplified and uncomplicated formula, yet fits the research objectives.

$$F = C\,.\,f\,.\,l\,.\left[\,m_R^l + \left(2\,m_G^l + m_L^l\right).\,cos\,\delta\,\right].\,g + H\,.m_L^l\,.g \quad \text{In N}$$

Equation 3-5

The required power calculates using the speed, taking to account the effect of time, eq3 and the eq4 for the required power to be installed considering the efficiency of electrical motors.

$$P_{Tr} = \frac{F}{1000}\,.\,v \qquad \text{In kW} \qquad \text{Equation 3-6}$$

$$P_M = P_{Tr}\,/\,\eta^{+} \qquad \text{In kW} \qquad \text{Equation 3-7}$$

3.2.1 Importance of index

Having a representative index is essential for any further assessment or optimization. A proper index enables us to understand the performance of a machine or process in a mine besides, it provides the opportunity to compare different mine's performances among varying ore types and deposit shapes. For belt conveyors, as a case in point, kWh per tonne has been used generally, while it represents energy efficiency in most cases, some conveyors' performance can still be misinterpreted. In diagram below, red triangles indicate the monthly energy consumption for selected numbers of conveyors by kWh/t and blue squares incorporate length and represent the same conveyors in the same month using kWh/t.km index. As diagram indicates, two conveyors' position change notably, means energy consumption may respectively be underestimated or overestimated.

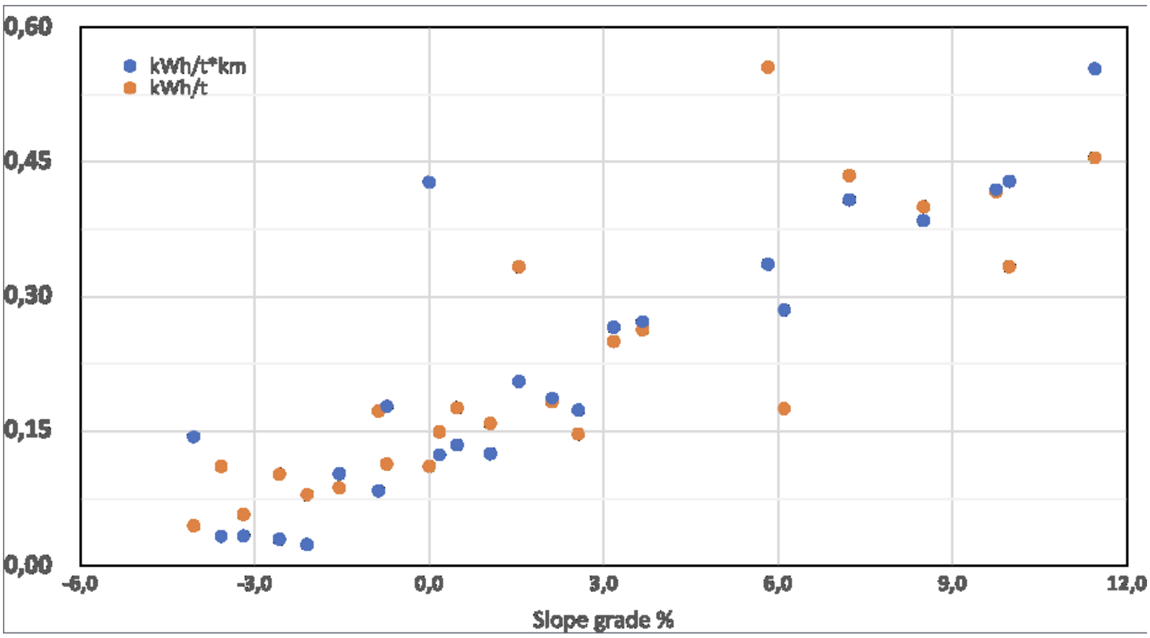


Figure 3-4 Energy Consumption comparison using kWh/t and kWh/t.km

3.2.2 The effect of dimension

It is important to note that dividing the energy consumption by length is appropriately functional and helpful for a conveyor with a length of more than 1000 meters. For having a more accurate comparison, conveyors less than 1000 meters the C in formula 3-3, the length coefficient, must be included. According to Figure 3-3 a 300 meters conveyor consumes 30% more energy per kilometre (or any other length unit) than a 1500 meters conveyor. The short conveyors consume more energy because of the higher secondary resistance, and they should not consider less productive. You can see this effect in the operational data, the conveyors from Hambach are divided into three categories. As chart below represents, conveyors less than 500 m, blue diamonds, have higher index value than the third category marked by red triangular, conveyors longer than 1000 m.

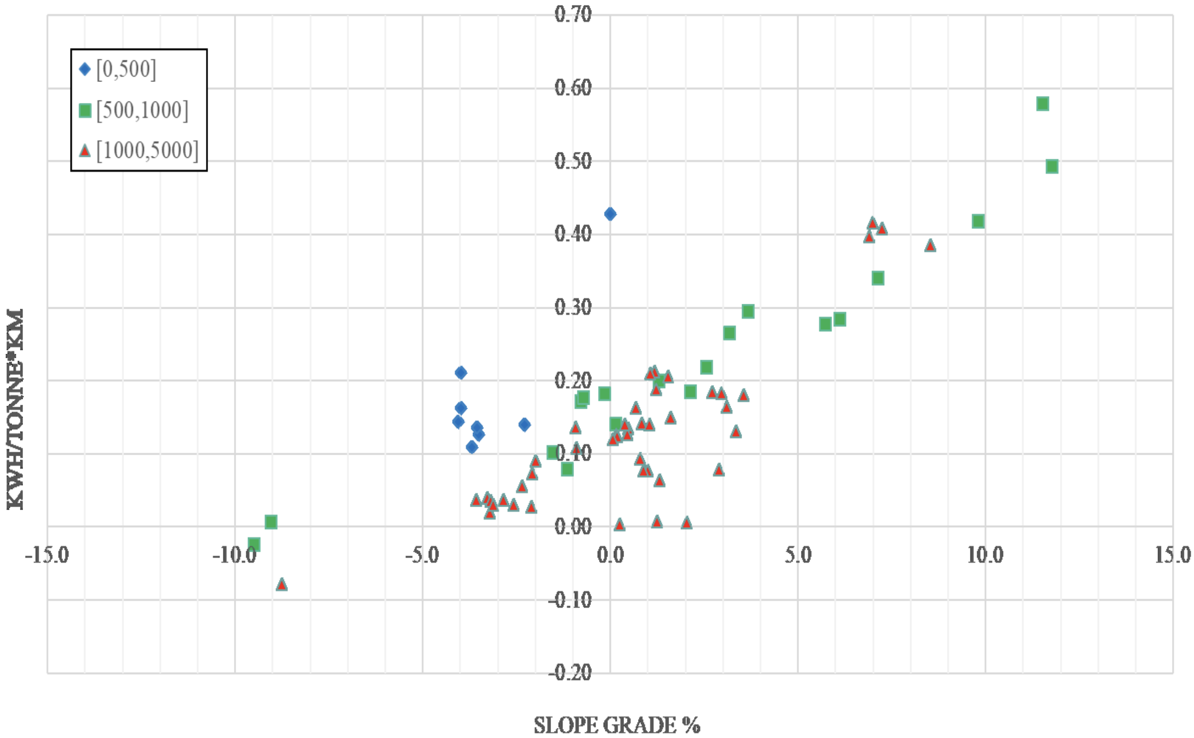


Figure 3-5 Energy consumption for different lengths.

3.2.3 Types of Material

Chart below includes both coal and waste conveyors, as the materials must be crushers to be transported by belt conveyors. The physical characteristics mainly affect the crushing units rather than conveyors. Even though some features like abrasiveness make a difference in terms of belt depreciation and lifetime, they do not affect when it comes to energy efficiency. The Chart shows the same energy consumption behaviour for waste material in blue triangles and coal in red squares.

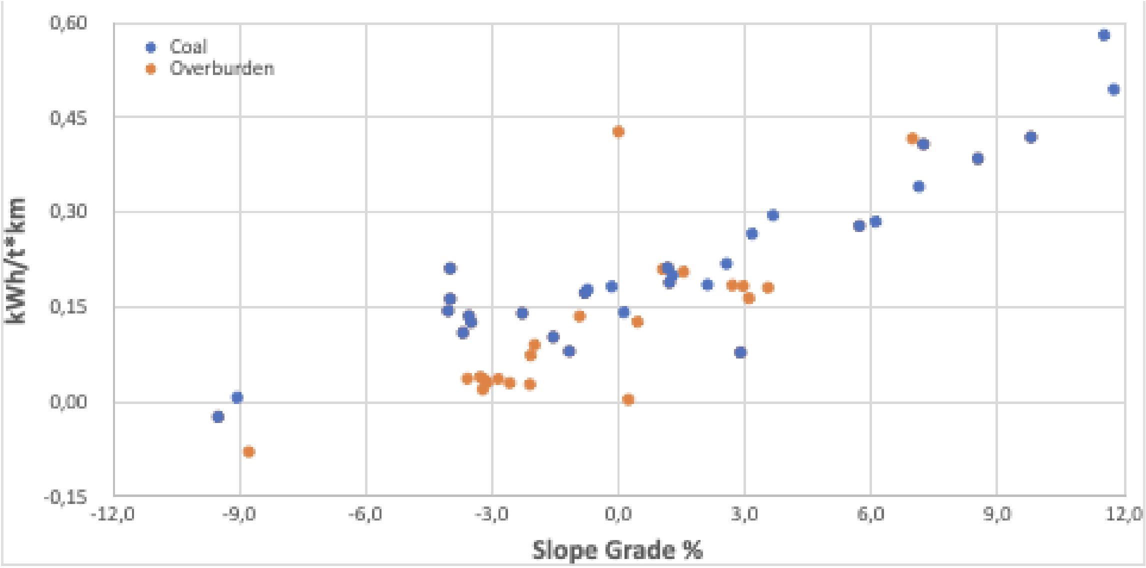


Figure 3-6 Consumed Energy for different types of material.

3.3 Utilization

In order to measure the actual energy consumption and to compare the results with the theoretical outcomes, the conveyors B701 and I10 were observed. B701 transports coal from an excavator to the distribution point, and I10 is a downhill waste conveyor with a negative slope that transports waste from the distribution point to a spreader. Table xx describes the information on B701 and I10. Two sets of data were used, the first set recording the hourly energy consumption for each conveyor, as shown in the energy columns in Table xx. The second set corresponds to the mass flow of material, for B701 is a bucket-wheel excavator production data, and for I10, data from spreader were used.

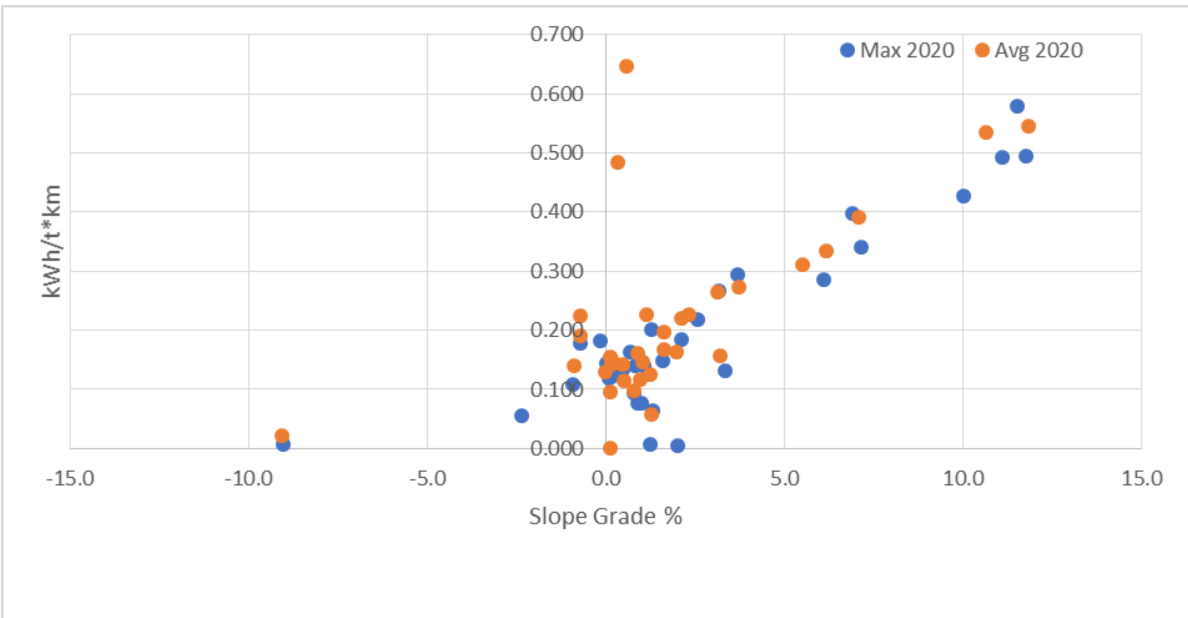


Figure 3-7 Average and best monthly energy consumption 2020.

3.4 Noise and vibration

Noise-induced hearing loss is a common problem in the mining industry. While considerations began in underground mining, the necessity to use heavy machinery in surface mining shows that hearing loss hazards occur in all mining areas. In fact, noise exposure above 90 dBA[1] over a period of time can cause noise-induced hearing loss (NIHL). The National Institute for Occupational Safety and Health (NIOSH) reported that over 70% of miners over 60 have a hearing loss greater than 25 dBA (NIOSH, National Occupational Research Agenda (NORA)., 1996). The 1996 analysis found that in the United States, nearly 90% of coal miners and 49% of metal/non-metal miners had hearing impairment by age 50 (Franks, 1996).

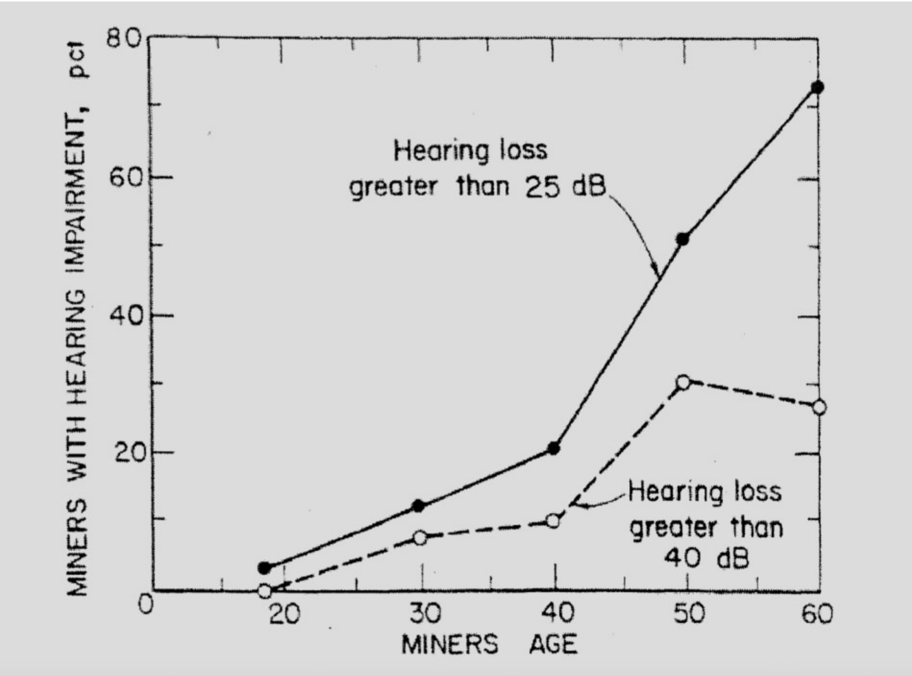


Figure 3-8 Hearing loss as a function of age (NIOSH, 1976)

For most machines in mining, the noise exposure of the machine operator is usually measured with the personal noise dosimeter and sound level meter . For belt conveyors in mining, they are usually monitored and inspected by supervisors, not operated. Therefore, the distance to noise sources depends on the conveyor configuration and inspection methods. Needless to say, the entire line does have to be shut down for maintenance and repair purposes. Another important factor to consider is the duration of noise exposure, as prolonged exposure to medium-high noise over a long period of time can cause severe hearing damage. In accordance with European standards, the table below shows the permissible duration for each sound level range.

[1] A-weighted sound levels

Table 3-1 Noise level equivalencies at various sound levels (EU Directive, 2003)

Noise Level dBA	Maximum Permissible Time of Noise Exposure, h
85	8
88	4
91	2
94	1
97	0.5
100	0.25

The noise emitted by conveyor belts is generated by the complex interaction of all conveyor belt components and is therefore difficult to specify. While it is possible to measure the noise of each individual part separately, it is important to determine the total noise level. The major noise-generating mechanisms in belt conveyors are the interaction of the idler and belt, and structure-borne noise. A coal belt conveyor with a capacity of 10000 tons per hour and a speed of five m/s has a noise level of 113 dBA to 119 dBA per 100 meters. Conveyor noise is usually described as a repetitive pulse. Analysis of the results of a conveyor belt with a capacity of 8000 tons per hour and a speed of five m/s at a distance of five meters shows strong amplitude modulation patterns, demonstrating the synchronized action of the idler pulley plays an important role as a noise-generating mechanism (Brown, 2015).

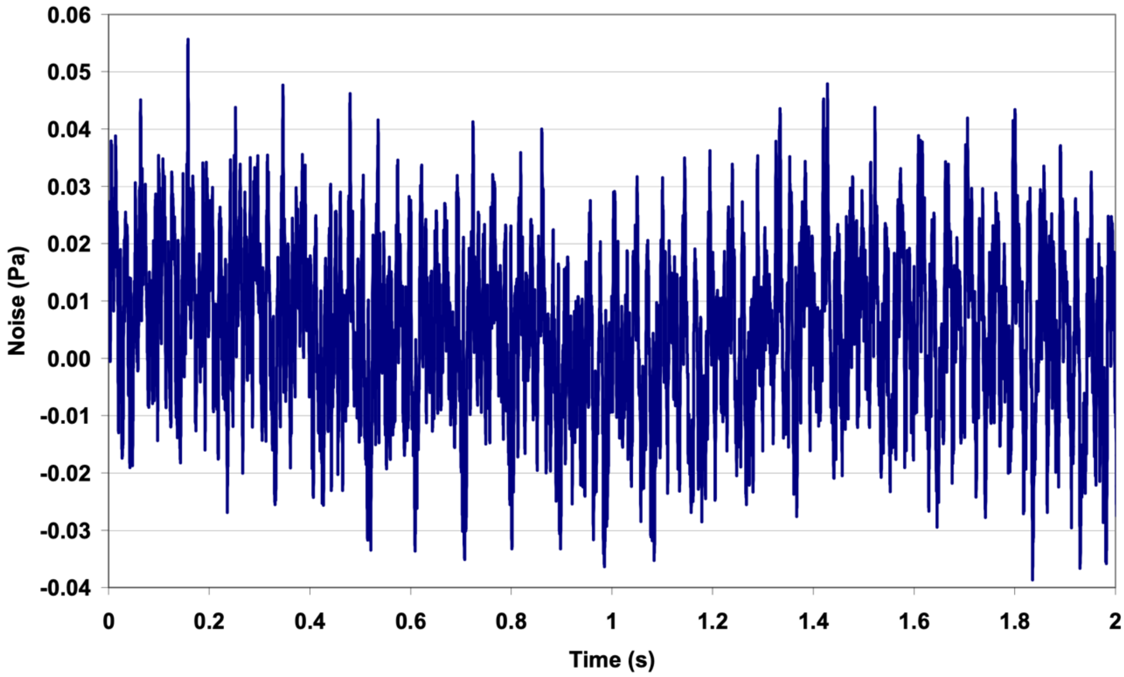


Figure 3-9 time history of conveyor noise (Brown, 2015).

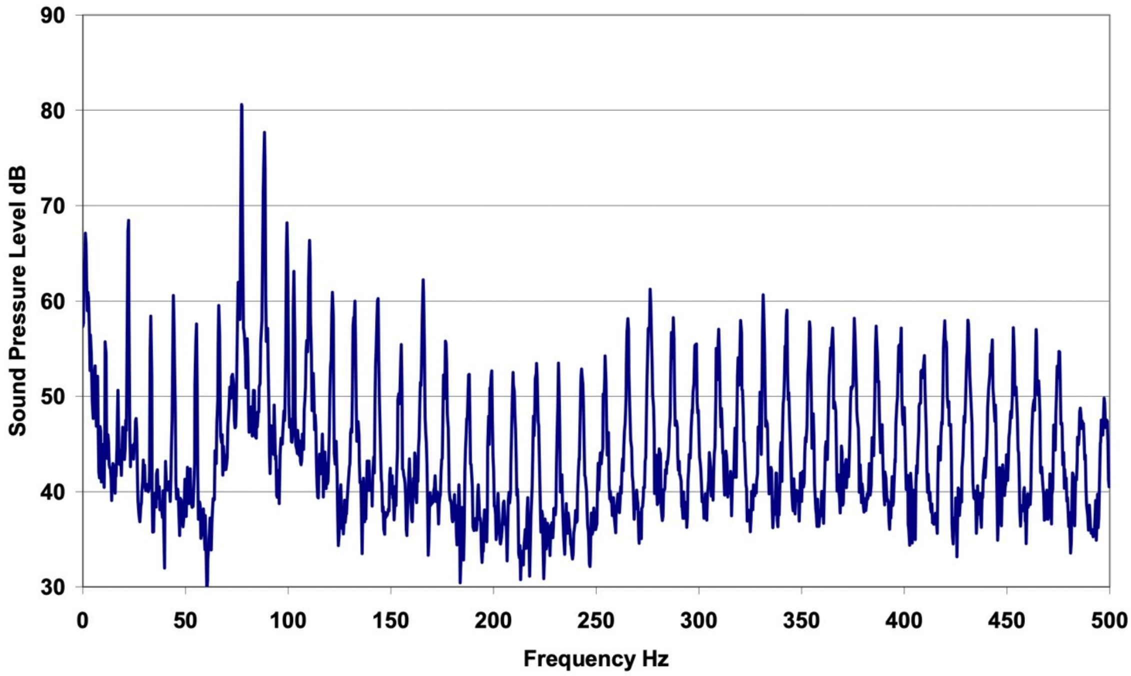


Figure 3-10 Typical Conveyor Narrow Band Noise Spectrum (Brown, 2015).

3.4.1 Vibration

Equipment vibration is a source of noise and can also cause premature equipment wear. Examples of equipment prone to vibration in transporting systems include compressors, electric motors, pulleys, conveyor belts, and diesel engines. Since conveyors are mainly composed of rapidly rotating parts, they are susceptible to vibration. Vibrations can occur directly when rotating parts are unbalanced, misaligned, loose or eccentric, or indirectly caused by another piece of the equipment. The vibration that is hazardous to health includes hand-arm vibration, which affects control of vibrating equipment, and whole-body vibration, which is associated with musculoskeletal problems, including back pain and degenerative disc disease. In accordance with ISO standard 5349-1 :200 vibration total value represents the sum of vibrations measured in three directions (X axis, Y axis and Z axis). The table below provides the durations equivalent to the exposure action levels and thresholds for both cases (Pelmear & Leong, 2002).

Table 3-2 Vibration magnitudes and durations equivalents (Pelmear & Leong, 2002).

Total exposure duration h	16	8	4	2	1	1/2
Average (r.m.s.) vibration magnitude to give daily exposure of 2.5 m/s^2	1.8	2.5	3.5	5	7	10
Average (r.m.s.) vibration magnitude to give daily exposure of 5 m/s^2	3.5	5	7	10	14	20

Vibration control can be iterative, involving incremental improvements as vibrating components are identified and controlled. Scheduled monitoring and control are essential to specify vibration sources. Isolating the disturbance from a radiating surface and reducing the response of the radiating surface can prevent chronic wear and mitigate health problems.

3.5 Dust emission and control

The airflow over the conveyor carries atmospheric aerosol particles. Carryback refers to material that sticks to the conveyor belt after the material is discharged at the head pulley. Airborne dust and uncontained material from carryback and spillage are the main sources of fugitive dust emissions.

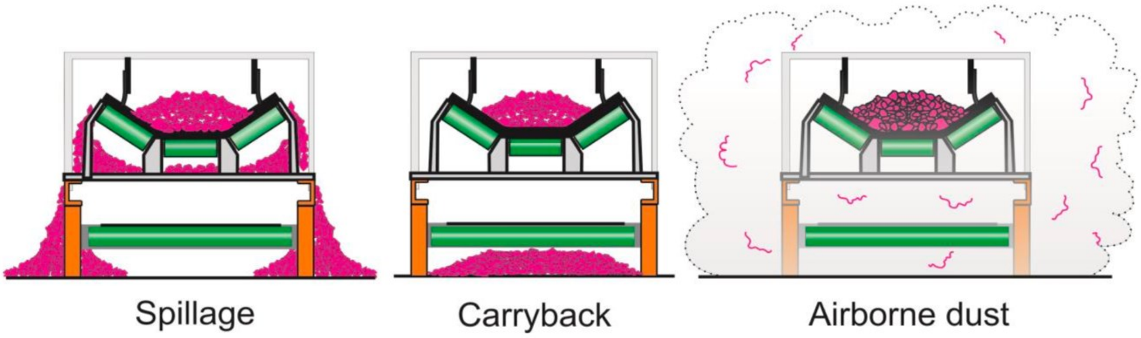


Figure 3-11 Sources of fugitive emissions (Cecala & al, 2019).

3.5.1 Containing the material

Properly loaded and evenly distributed material on the belt provides stable conveying and reduces spillage either at transfer points or along the line. The vertical drop height of the material at the loading and transfer points is related to dust emission and results in air entertainment into the material. In addition, high drop heights result in belt damage by heavier lumps impacts and also cause material rebound and spillage. Minimizing drop heights and absorbing material impact using impacts cradles and rock boxes containing material on the belt and improve dust control.

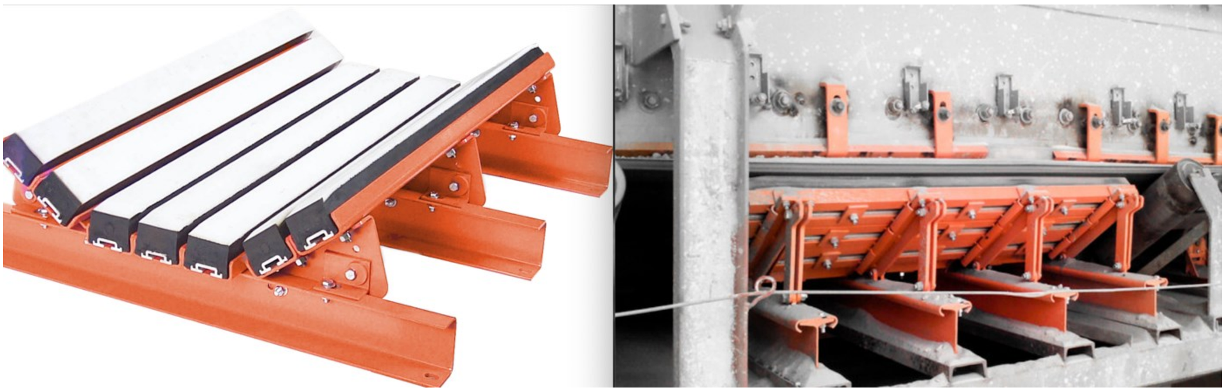


Figure 3-12 Impact cradles (Martin Engineering, 2022).

An impact cradle system installed under the receiving belt absorbs most of the material impact on the belt during loading. Impact cradles use low-friction bars under the belt to absorb the rebound and protect the receiving belt. A chute design with rock boxes split drop heights and, accordingly, lower speed and momentum of material. The use of rock boxes not only reduce the wear of belt and idlers it also prevents major air induction and dust emission (Martin Engineering, 2022).

3.5.2 Controlling the airflow

When the material discharges through the chutes, the air is entrained (venturi effect) and can thus pressurize the airflow. Due to the increased air velocity, more particles are carried, and more dust is stirred up. Although complete dust elimination is not practicable, regulating air velocity and material flow can improve dust control. A special chute design can help reduce material and air turbulence generation by keeping the material in a narrow stream. An enclosure at the end of the feeding conveyor with staggered rubber curtains reduces air entrainment and minimizes airflow at the entrance. The use of multiple intermediate curtains extends the path of the air, called the stilling zone, and allows additional particles to settle out of the air and return to the ore. Finally, enclosure by robber curtains at the exit point keeps airborne dust inside the transfer point (Cecala & al, 2019).

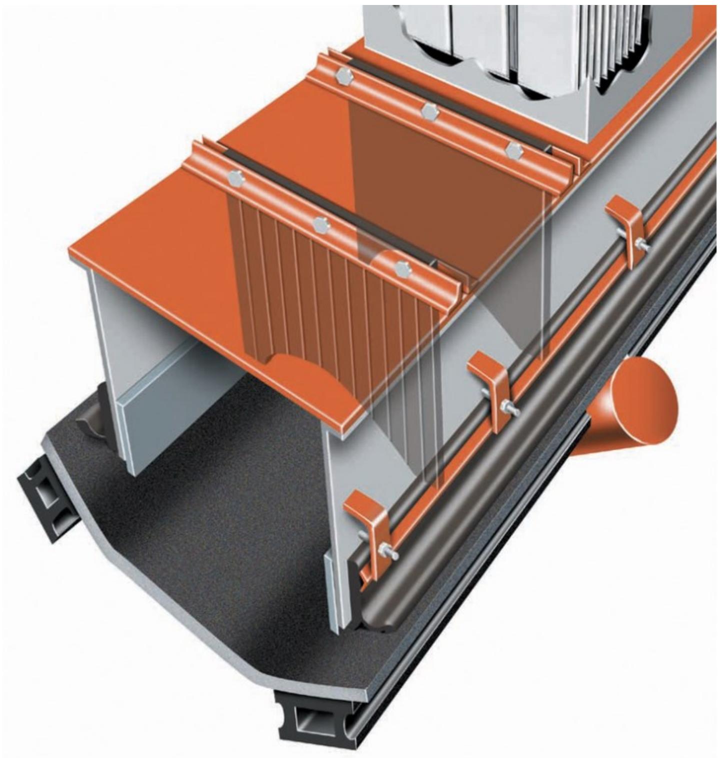


Figure 3-13 Entry and intermediate dust curtains (Martin Engineering, 2022).

3.5.3 Wet spraying system

Water spray systems are the most common and effective method of dust control in mining operations. In general, wet spraying involves two stages: first, the prevention of dust migration, and second, the suppression of the stirred-up dust particles.

In the prevention stage, the cohesion of the material particles is increased by spraying the material directly. The aim is to increase the moisture content of the material and increase the weight of the particles, reducing their ability to disperse in the air. In suppression, spraying the dust cloud removes the particles suspended in the air and increases their mass, causing the particles to agglomerate and fall out of the air back onto the material on the conveyor or transfer points.

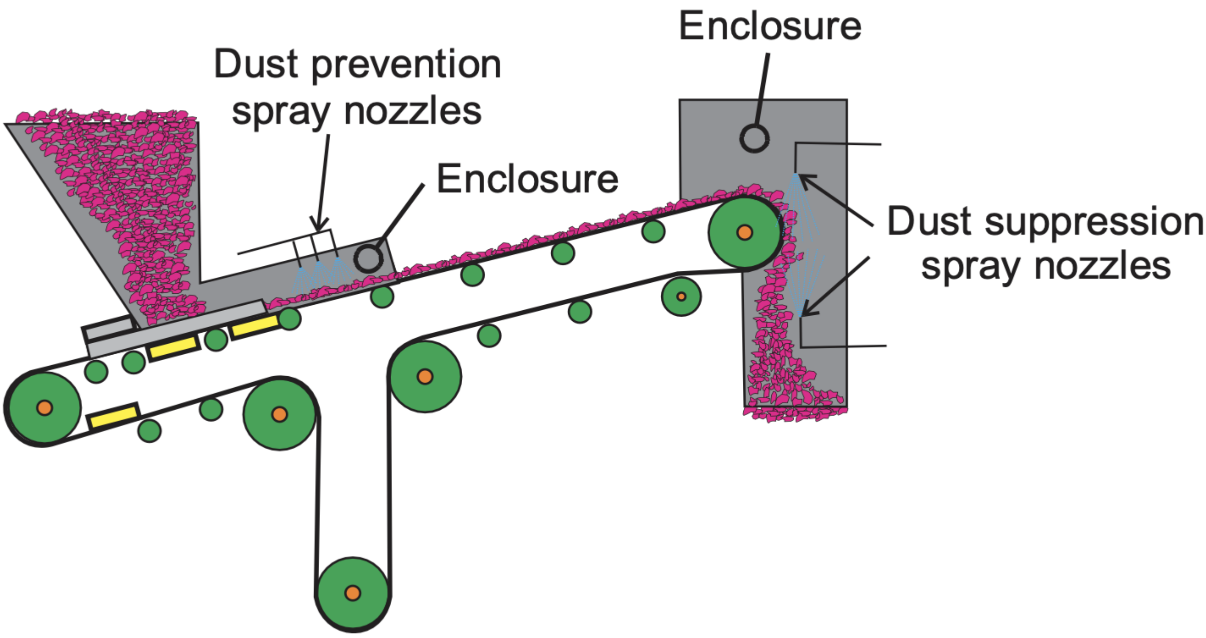


Figure 3-14 Dust prevention and suppression application (Cecala & al, 2019).

Wet spray systems should be designed with consideration of ore type, application point, and weather conditions. System performance depends on effective use of moisture, careful positioning of nozzles, control of droplet size, spray pattern, and nozzle type. Excessive water use will cause problems such as belt slippage, back drag, and chute clogging (Cecala & al, 2019).

4 Environmental assessment of mining trucks

4.1 Introduction

Trucks have been used in open-pit metal mines as a major way of transporting for past decades. They are divided into classes according to their payload capacity. The size of the trucks and the classes gradually increases over the years, as the use of higher classes requires fewer operators for the same production volume, and the payload to empty weight ratio is slightly better. The size of a deposit determines its production per year (PPY); the larger the deposit, the greater the investment required.

A secure PPY and the corresponding investment rate of return guarantee the profitability of the project and reduce the financial risks. In the context of transportation, the truck's payload capacity is associated with the time value of money. It is not economically feasible to achieve the PPY goal of a mine designed for high-class trucks with mid-class or low-class options, as both capital and operating costs will increase. Another factor is the operational aspect: a large number of medium and small trucks will increase the volume of traffic in the mine and cause more stops and waiting times.

Another reason for having bigger trucks is that the bench height is designed according to deposit size and PPY; therefore, the loading units must be selected so that they can reach and load the entire height of the bench. The larger loading unit means a larger bucket capacity, whether it is a shovel, loader, or dragline. Generally, it is recommended to load a truck with 3 to 5 passes and traditionally assumes loading units are more expensive; therefore, it is better to avoid the fleet management plans that keep them idle. For having a more accurate passes number, a cost model for a shovel, for example, and truck can be used, and the number should be the closest integer number to the ratio of the total cost of the shovel to the total cost of the truck.

Finally, the use of lower-class trucks means that the shovel is idle and has to wait more frequently for the loaded trucks to leave the loading area and for the empty trucks to change. In addition, a larger number of trucks and drivers in the pit leads to more accidents and human error, which in turn increases safety risks. All of the above factors favor the use of high-class trucks; on the other hand, there are also some unfavorable factors. The first, and perhaps most important, is the technical factor; it is a challenge to manufacture and maintain trucks with payloads of over 400 tons at feasible scales.

The other factor is related to the design of the pit and the roads. Large trucks require broader routes, which means that the pit must be wider, and the overall slope of the mine decreases. This increases the stripping ratio and increases the payback time of the mine in the early stages. Driving a high-class truck is also challenging: visibility is more limited, maneuverability is more restricted, and the driver needs more skills to use the braking systems properly. It is also more difficult to find and train qualified drivers in remote areas. These are the conventional factors and considerations for selecting trucks, and they do not take into account environmental concerns in practice, nor is energy efficiency a determinant factor.

Fuel consumption is part of operating costs, and as supply risk and fuel prices have historically remained low, managing and optimizing energy efficiency plays a secondary role. As global warming has never been as crucial as it is today, and energy supply has never been as critical as it is now, it is more important than ever to first quantify the environmental impact and then propose solutions for efficient management. This chapter introduces the operating parameters of trucks, explains the problem of typical calculation approaches for energy management, discusses common environmental impacts, and presents a new method for evaluating greenhouse gas emissions.

4.2 Operating parameters

4.2.1 Payload policy

The nominal payload of rigid frame trucks ranges from 35 to 350 metric tonnes, while they are generally categorized into five classes 90t, 135t, 180t, 220t, and more than 290t class or ultra-class. Each of these classes required a specific road width that affects the stripping ratio and mine design. Trucks should usually be loaded with three to five passes of loading machine; otherwise, they need to stay longer for loading and operate in longer cycles, which means more trucks and higher capital costs are required to operate. Truck manufacturers recommend similar payload guidelines for optimal loading of dump trucks; for example, Caterpillar suggests the 10/10/20 payload policy. The policy states that "no more than 10% of loads may exceed 10% over the target payload and no loads may exceed 20% of the target payload" (Caterpillar, A Reference Guide to Mining Machine Applications, 2009).

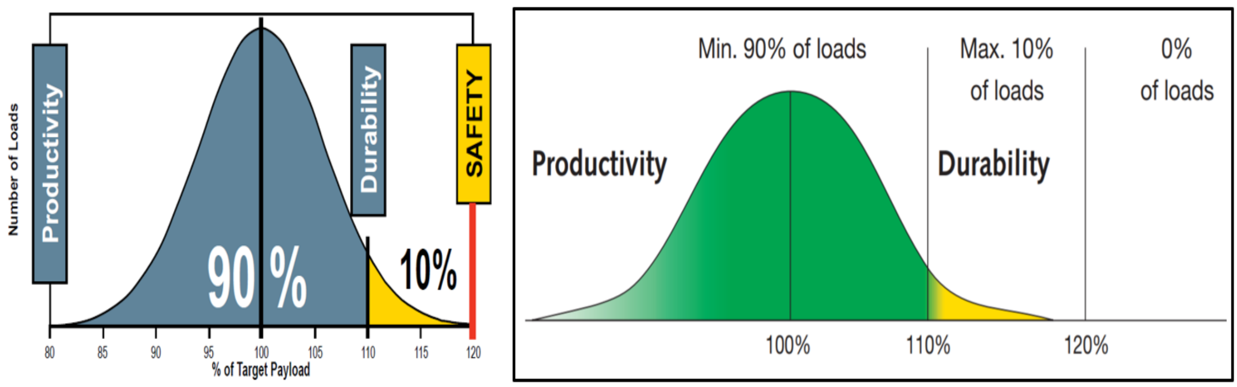


Figure 4-1 10/10/20 payload policy diagram from (Caterpillar, 2009) left, and Komatsu (Kumatsu, 2021) right.

4.2.2 Truck's Operating Modes

As mentioned before, estimating trucks' fuel consumption is essential not only for cost calculation also for assessing GHG emissions. Many parameters are bearing effect on fleets' efficiency, like mine plan, dumpsites design, vehicle operating parameters including engine efficiency, and transmission shift patterns.

However, if one assumes loading and discharge locations as governing parameters, then payload, speed, and cycle time could be regarded as the subject of calculation and optimization for mine road profiles. By measuring the dispensed amount and time between each refuel process, it is possible to calculate consumed fuel for the entire fleet, or in some cases, for each truck. The consumed amount of fuel depends on duration of cycle time, the ratio between each cycle mode's time, and how truck drives during moving time, particularly when it is loaded. **Error! Reference source not found.** shows cycle modes and approximate times from Caterpillar mining reference guide (Caterpillar, A Reference Guide to Mining Machine Applications, 2009).

Table 4-1 Cycle modes and required times (Caterpillar, 2009).

Cycle Mode	Required Time
Standing in line for loading *	Depends on fleet management
Truck exchange	54 seconds
Loading time (3-5 passes)	37 seconds for each pass
Moving loaded	Depends on hauling distance, trucks specification, road condition
Standing in line for unloading and unloading	Depends on fleet management
Moving unloaded	Depends on hauling distance, trucks specification, road condition

* Fixed truck allocation assumed, without dispatching system.

Since fuel consumption rate is the same when the truck stands idle, whether in lines or for loading, the cycle time could be simplified into three modes: idling, moving loaded, moving unloaded. As an example, for a coal surface mine all loading, stopped loading and stopped time could be regarded as idling time.

In an open-pit operation, trucks move uphill with 7-10 % slope grade when they are loaded, this mode accounts for 70-80 % of total cycle fuel consumption. After unloading, trucks come back unloaded, the best case is when the roads have 3 % of slope grade, which equals the vehicle's rolling resistance on the flat road, then fuel consumption rate is equal to idling mode, though they usually come back through the same road with 7-10% slope grade this means they need energy to limit their speed, this mode so-called retarding and some electric trucks can save energy during this mode. Idling fuel consumption rate among all different classes is about 10-15 litter per hour (Caterpillar, The 48th Caterpillar® Performance Handbook, 2018).

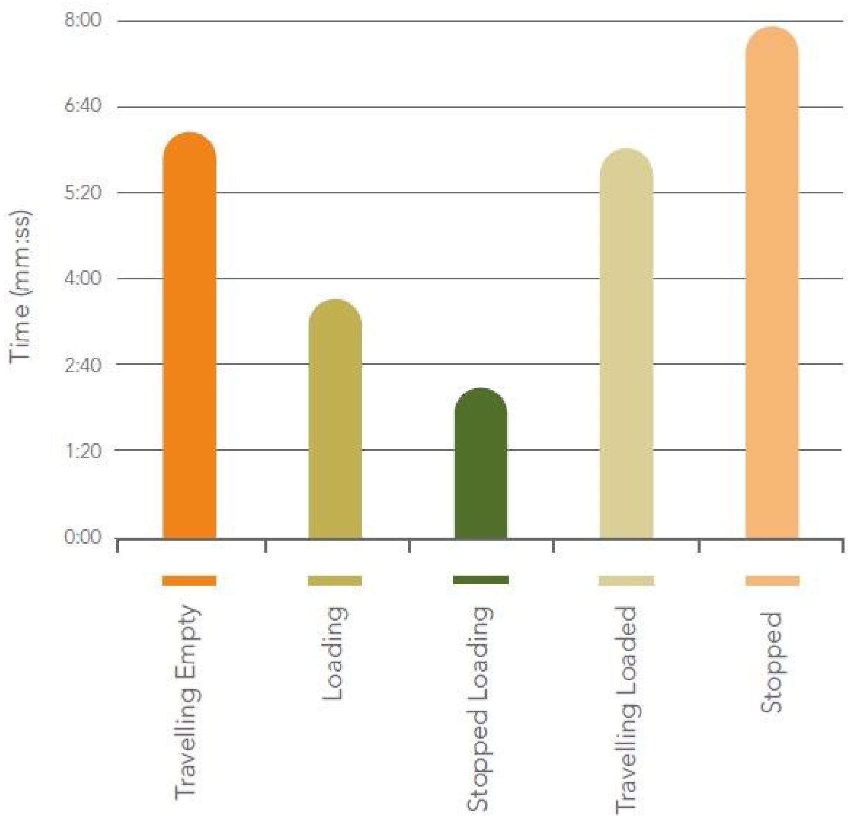


Figure 4-2 Caterpillar 777F,90t operating modes within a payload cycle (Australian Government, 2010).

4.2.3 Effective grade

Calculating a fleet's cycle time starts with effective grade calculations, which is equal to the sum of slope grade and rolling resistance.

$$\text{Effective grade} = \text{Slope grade (\%)} + \text{Rolling resistance (\%)} \qquad \text{Equation 4-1}$$

Slope grade depends on the distance between shovels and crushers, stockpiles, and dumpsites both vertically and horizontally. Steeper slope grade means shorter transporting distance, on the other hand, trucks have operational constraints that limit them to drive loaded up to 15 % slope grade. Working on steep roads requires more engine load and more frequent maintenance and repair alongside increased fuel consumption rate; therefore, roads are designed with 7-10 % slope grade, while 8 % is usually recommended. Rolling resistance highly depends on road condition and usually states in slope grade equivalent, below is the table for roads' rolling resistances.

Table 4-2 Rolling resistance for different road conditions (Caterpillar, 2018).

Road Condition	Rolling Resistance (%)
Hard, well-maintained, permanent haul road	**1.2**
Well-maintained road with little flexing	**2.5**
Road with 25 mm (1 in) tire penetration	**4**
Road with 50 mm (2 in) tire penetration	**5**
Road with 100 mm (4 in) tire penetration	**8**
Road with 200 mm (8 in) tire penetration	**14**

4.2.4 Fuel consumption rate based on time

Mining companies keep records for their production, based on time, for example, per shift or day. After dividing consumed fuel by tonnage of moved material, the result as an indicator usually uses to show fleet's efficiency. However, this indicator could not represent the efficiency properly even if divided by hauling distance. Although the indicator uses real data still does not reflect how trucks operate during their cycle time. The next example tries to explain this issue:

For the sake of calculation, we assume there is an open-pit mine with an average of 3 km of hauling distance, and we use CAT 785D to transport material.

Table 4-3 Caterpillar rigid frame truck data (Caterpillar, 2018).

TRUCK'S MODELL	CATERPILLAR 785D
Gross machine weight	250 t
Nominal Payload	140 t
Net Power	1005 kW
12 Cylinders and Six Speed Gearbox	
Average Slope grade	10 %
Average haulage distance	3 km
Approximate speed for loaded mode	12 km/h
Approximate speed for loaded mode	30 km/h

The first step for computing cycle time is determining trucks' speed on the roads; the manufacturer's catalogue is used to do this (Caterpillar, The 48th Caterpillar® Performance Handbook, 2018).

The diagram for cat 785D, start with Gross Machine Weight (dead load plus payload), which is 250 tonnes. Move vertically until the effective grade line. This example's effective grade is 13 %, 10 % for road slop grade, and 3 % for rolling resistance. From this point, move horizontally until intersecting one (in some cases two) curve to define optimal gear. Finally, read speed vertically down from the bottom axis (in this case around 12 km/h). For unloaded mode, the procedure is similar, the Gross Machine Weight is 100 tonnes, and the effective grade is 7 %, -10 % slope grade plus 3 % of rolling resistance. Based on the diagram this model could operate at 37 km/h, but for safety reasons 30km/h is considered.

By optimistic approach, regard these speeds as the average for the entire 3 km road regardless of any stop, acceleration, and deceleration. The table below showed three scenarios for cycle time and used the CAT performance handbook for hourly fuel consumption (Caterpillar, The 48th Caterpillar® Performance Handbook, 2018).

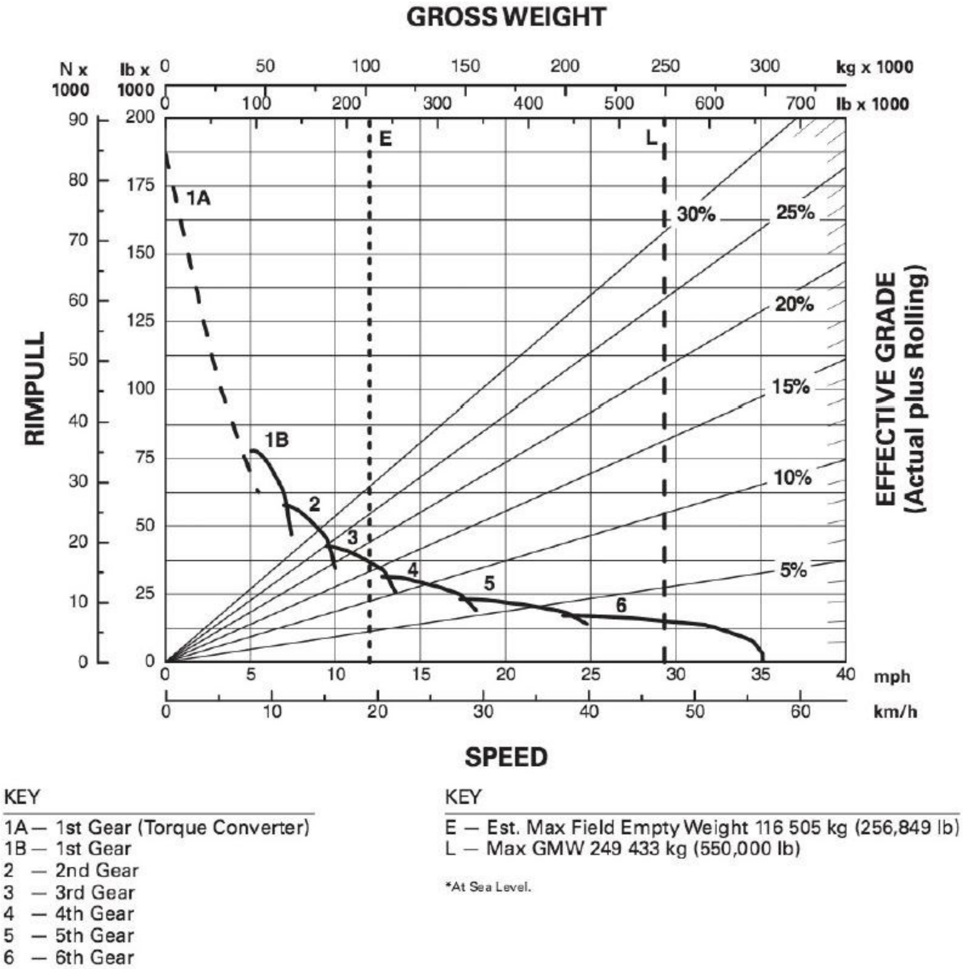


Figure 4-3 Rimpull-Speed-Gradeability Chart (Caterpillar, 2018).

Low: Continuous operation at an average gross weight less than recommended. Excellent haul roads. No overloading, low load factor.

Medium: Continuous operation at an average gross weight approaching recommended. Minimal overloading, good haul roads, moderate load factor.

High: Continuous operation at or above maximum recommended gross weight. Overloading, poor haul roads, high load factor.

Table 4-4 Fuel consumption rate lit/h (Caterpillar, 2018).

MINING & OFF-HIGHWAY TRUCKS

Model	Low		Medium		High	
	liter	U.S. gal	liter	U.S. gal	liter	U.S. gal
770G[1,2]	12.5-16.5	**3.3-4.3**	16.5-21.5	**4.3-5.6**	21.5-27.0	**5.6-7.1**
772G[1,2]	14.5-18.0	**3.8-4.7**	18.0-24.0	**4.7-6.3**	24.0-32.0	**6.3-8.4**
773E[3]	—	—	—	—	—	—
773G[2]	18.5-28.0	**4.9-7.3**	28.0-35.0	**7.3-9.2**	35.0-43.0	**9.2-11.3**
775G[2]	24.0-28.5	**6.3-7.5**	28.5-36.0	**7.5-9.4**	36.0-47.0	**9.4-12.3**
777E[3]	—	—	—	—	—	—
777G[2]	33.0-46.0	**8.7-12.1**	46.0-57.0	**12.1-15.0**	56.5-72.5	**14.8-19.0**
785C 1450 HP[4]	53.7-80.6	**14.2-21.3**	80.6-107.5	**21.3-28.4**	107.5-134.4	**28.4-35.5**
785D 1450 HP[5]	54.2-81.4	**14.3-21.5**	81.4-108.5	**21.5-28.7**	108.5-135.6	**28.7-35.8**
789D 1900 HP[4]	70.6-105.9	**18.7-28.0**	105.9-141.2	**28.0-37.3**	141.2-176.5	**37.3-46.6**
789D 2100 HP[4]	74.9-112.4	**19.8-29.7**	112.4-149.9	**29.7-39.6**	149.9-187.4	**39.6-49.5**
789D 2100 HP[5]	79.7-119.5	**21.1-31.6**	119.5-159.3	**31.6-42.1**	159.3-199.1	**42.1-52.6**
793D 2160 HP[4]	82.2-123.3	**21.7-32.6**	123.3-164.4	**32.6-43.4**	164.4-205.5	**43.4-54.3**
793D 2270 HP[4]	86.0-129.0	**22.7-34.1**	129.0-172.0	**34.1-45.4**	172.0-215.0	**45.4-56.8**
793D 2415 HP[4]	90.8-136.2	**24.0-36.0**	136.2-181.6	**36.0-48.0**	181.6-227.0	**48.0-60.0**
793F 2270 HP[4]	82.4-123.7	**21.8-32.7**	123.7-164.9	**32.7-43.6**	164.9-206.1	**43.6-54.5**
793F 2270 HP[5]	87.2-130.8	**23.0-34.6**	130.8-174.4	**34.6-46.1**	174.4-218.0	**46.1-57.6**
793F 2650 HP[4]	95.0-142.5	**25.1-37.6**	142.5-190.0	**37.6-50.2**	190.0-237.6	**50.2-62.8**
793F 2650 HP[5]	96.8-145.2	**25.6-38.4**	145.2-193.6	**38.4-51.1**	193.6-242.0	**51.1-63.9**
794 AC 2750 HP[4]	99.9-149.9	**26.4-39.6**	151.9-199.9	**40.1-52.8**	199.9-249.9	**52.8-66.0**
794 AC 3100 HP[4]	111.5-167.3	**29.5-44.2**	169.5-223.0	**44.8-58.9**	223.0-278.8	**58.9-73.7**
794 AC 3500 HP[4]	126.4-189.7	**33.4-50.1**	192.2-252.9	**50.8-66.8**	252.9-316.1	**66.8-83.5**
794 AC 2750 HP[5]	100.5-150.7	**26.5-39.8**	152.7-200.9	**40.3-53.1**	200.9-251.2	**53.1-66.3**
794 AC 3100 HP[5]	112.8-169.1	**29.8-44.7**	171.4-225.5	**45.3-59.6**	225.5-281.9	**59.6-74.5**
794 AC 3500 HP[5]	126.1-189.2	**33.3-50.0**	191.7-252.2	**50.6-66.6**	252.2-315.3	**66.6-83.3**
793F HAA 2650 HP[4]	90.7-136.0	**24.0-35.9**	136.0-181.4	**35.9-47.9**	181.4-226.7	**47.9-59.9**
795F 3100 HP[4]	113.0-169.5	**29.9-44.8**	169.5-226.1	**44.8-59.7**	226.1-282.6	**59.7-74.7**
795F 3100 HP[5]	113.1-169.7	**29.9-44.8**	169.7-226.3	**44.8-59.8**	226.3-282.8	**59.8-74.7**
795F 3400 HP[4]	124.4-186.6	**32.9-49.3**	186.6-248.8	**49.3-65.7**	248.8-311.0	**65.7-82.2**
795F 3400 HP[5]	124.4-186.6	**32.9-49.3**	186.6-248.8	**49.3-65.7**	248.8-311.0	**65.7-82.2**
795F HAA 3400 HP[4]	121.4-182.1	**32.1-48.1**	182.1-242.8	**48.1-64.1**	242.8-303.6	**64.1-80.2**
797B 3550 HP[4]	133.5-200.2	**35.3-52.9**	200.2-266.9	**52.9-70.5**	266.9-333.6	**70.5-88.1**
797F 3550 HP[4]	126.0-189.1	**33.3-50.0**	189.1-252.1	**50.0-66.6**	252.1-315.1	**66.6-83.2**
797F 3550 HP[5]	131.5-197.3	**34.7-52.1**	197.3-263.0	**52.1-69.5**	263.0-328.8	**69.5-86.9**
797F 4000 HP[4]	143.1-214.6	**37.8-56.7**	214.6-286.2	**56.7-75.6**	286.2-357.7	**75.6-94.5**
797F 4000 HP[5]	146.3-219.4	**38.7-58.0**	219.4-292.5	**58.0-77.3**	292.5-365.7	**77.3-96.6**
797F HAA 4000 HP[5]	147.9-221.8	**39.1-58.6**	221.8-295.8	**58.6-78.2**	295.8-369.7	**78.2-97.7**

;ea Level data only, power and fuel will change with altitude.
/alue is product of load factor and rated power fuel burn, actual fuel burn will vary with engine speed and load.
t is recommended to use FPC (Fleet Productivity and Cost analysis) software for the best cycle and hourly fuel burn calculations.
IAA – High Altitude Engine Arrangement
Machine requires the use of DEF with a consumption rate approximately 2-3% of diesel fuels.
Fuel consumption based off machines reporting into VisionLink®.
VisionLink fuel consumption not available by the time of publication, please consult the factory for information.
Meets Tier 1 equivalent emission standards.
Meets Tier 2 equivalent emission standards.
Machine requires the use of DEF with a consumption rate approximately 2-3% of diesel fuel.

Base on this table, the medium column corresponds to an average of 40 % of the engine load factor. The same approach commonly uses for cost models, which are fundamentally calculating base on hour. Aforementioned time based approaches deal with the last row in the Table 4-4 Fuel consumption rate lit/h .hourly fuel rate, and it changes significantly, almost 18 and 30 %, while the needed amount of fuel is relatively constant and reduces only 2 and 4 %. Even though, a 20% change in cycle time directly affects the overall performance yet does not significantly affect fuel consumption. When the trucks are idling, the fuel consumption rate is only ten lit per hour, while when they are moving (whether loaded or unloaded), they need 100 litre per hour.

Table 4-5 Fuel rates for three cases with different idling times.

	Case 1	**Case 2**	**Case 3**
Cycle time (min)	20	25	30
Loaded and unloaded time (min)	20	20	20
Average Loaded and unloaded fuel use rate (lit/h) *	100	100	100
Idling time (lit/h)	0	5	10
Idling fuel rate (lit/h)	10	10	10
Cycle fuel rate (lit)	33,33	34,16	34,99
Overall fuel rate (lit/h)	100	82	69,99

*The measured consumption is higher, the manufacturer approximate figures assumed to assess the time base approach accuracy (Caterpillar, 2018).

4.3 Energy efficiency

4.3.1 Mathematical Model

As shown in the example, the time base calculation is highly sensitive to the estimation of the cycle time and the fuel consumption rate. Even so, the energy needed to move an object is a function of mass and distance.

$$W_{(J)} = F.d_{(N.m)}$$ Equation 4-2

$$W = (ma + mgh)\ .d$$ Equation 4-3

$$W = \left(\frac{1}{2}mv^2 + mgh + \sum f\right).d$$ Equation 4-4

f: frictional forces

The variables are mass and displacement. Please note that v changes during the move are not consider. Despite its nature, the formula treats speed as a time-independent variable. It is similar to calculating the required energy to move an object vertically; in the better word changing Gravitational Potential Energy, the formula does not consider how fast the object moves.

For calculating power, time takes into account. Also, efficiency factor is needed for the final estimation.

$$P\ (\mathrm{W\ or\ kW}) = \frac{W\ (\mathrm{J\ or\ kWh})}{t\ (\mathrm{s\ or\ h})} \qquad \text{Equation 4-5}$$

Returning to the truck example, it should be remembered that if the possible acceleration and deceleration neglect, the speed of a truck class in loaded mode does not change significantly in the standard slope range. This is because the throttle valve is open 100% so that the engine can achieve its maximum performance, and the truck drives at the achievable speed specified by the manufacturer.

In other words, most of the energy consumption of trucks could be estimated to be almost time-independent. However, the number of trucks, CAPEX, and OPEX highly depend on cycle time and speed.

4.3.2 Fuel consumption rate:

With a simple linear equation, it is possible to calculate the hourly fuel consumption for different engine load factor regardless of the fuel density, (Kecojevic & Komljenovic, 2010). Where P is known as nominal engine power, and C is the only parameter that needs to be calculated.

$$FC = P \times C \times LF \qquad \text{Equation 4-6}$$

FC: Fuel Consumption (l/h)

C: Conversion Factor (l/kWh)

LF: Engine Load Factor (%)

P: Engine Power (kW)

$$C = \frac{FC}{P \times LF} \qquad \text{Equation 4-7}$$

4.3.3 Calculating the conversion factor based on performance data

The following table shows the trend line between Engine power and Fuel rate based on Caterpillar's Table 4-4. The linear behavior indicates the C factor is constant through models, and once it was calculated for one specific load factor, it could be set as a benchmark for other load factor values.

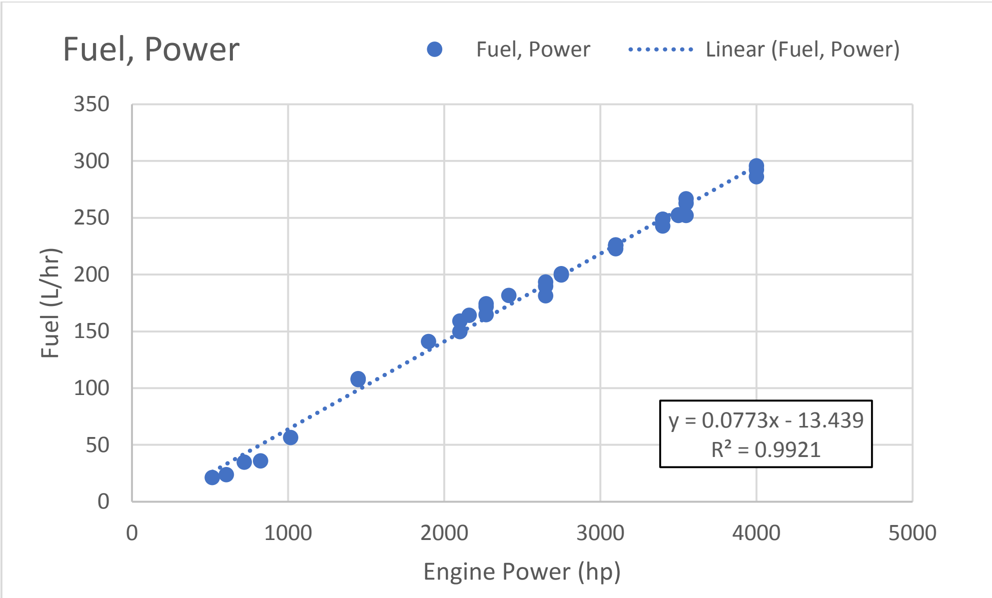


Figure 4-4 Fuel consumption rate and motor power trend (40% engine load factor).

The conversion factor is calculated based on the fuel consumption rate of the load factor of 40% using the equation 4-7 .Regardless of the truck model and payload, the 0.25 value for conversion factor values is relatively constant as trucks have more or less the same configuration. The value 0.25 is assumed for benchmarking and the next calculations, so the formula for fuel consumption is converted as follows:

$$FC = P \times 0.74 \times 0.25 \times LF \quad \text{Equation 4-8}$$

Table 4-6 Caterpillar trucks model and configuration.

Model	Target Payload (t)	Motor Power (hp)	High Fuel Rate (L/hr)	Motor Power (kW)	Conversion Factor	GMW (t)
785D 1450	136	1450	108.5	1081.27	0.25	250
789D 1900	185	1900	141.2	1416.83	0.25	320
789D 2100	185	2100	159.3	1565.97	0.25	320
793D 2160	227	2160	164.4	1610.71	0.26	380
793D 2415	227	2415	181.6	1800.87	0.25	380
794 AC 2750	291	2750	199.9	2050.68	0.24	508
794 AC 3500	291	3500	252.2	2609.95	0.24	508
795F 3100	315	3100	226.1	2311.67	0.24	568
795F 3400	315	3400	248.8	2535.38	0.25	568
795F HAA 3400	315	3400	242.8	2535.38	0.24	568
797B 3550	363	3550	266.9	2647.24	0.25	637
797F HAA 4000	363	4000	295.8	2982.80	0.25	637

As an example the 789 D conversion factor is 0.25 so if load factor changes to 30 % the calculated value using the equation 4-7 the Table 4-6 from manufacturer shows 123.3 for mentioned load factor.

$$C\ CAT\ 789D = 141.2 \div (1900 \times 0.74 \times 0.40) = 0.25$$

$$FC\ 793D = 2160 \times 0.74 \times 0.25 \times 0.30 = 123.07\ l/h\ (123.3)$$

Using formula for the biggest truck 50 % engine load factors shows on 0.3 difference.

$$FC\ 797F = 4000 \times 0.74 \times 0.25 \times 0.5 = 370\ l/h\ (369.7)$$

Even though the formula is based on the caterpillar's date and performance, it is also valid for other manufacturers. The following table and diagram are from Liebherr for mainly ultra-class trucks (almost more than 200 tons of payload). FC 1 is provided by Liebherr, and FC2 is calculated using the formula above, 0.25 conversion factor. The last column shows the difference.

Table 4-7 Fuel consumption rate for 100 % load factor from manufacturer and calculated by formula

Examples from Liebherr (LF= 100):			
P (hp)	FC_1	FC_2	Dif %
2500	455	462	1.5
2900	522	536	2.6
3000	547	555	1.5
3500	617	647	4.8
3650	640	675	5.4

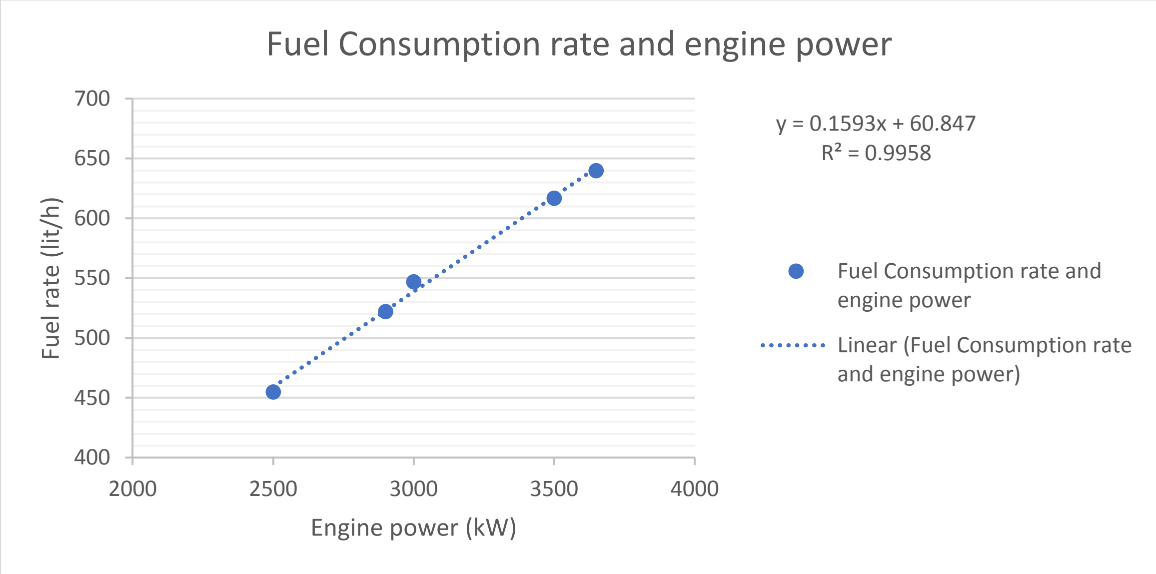


Figure 4-5 Fuel consumption rate and engine power at 100% load factor (Kecojevic & Komljenovic, 2010).

The formula below comes from the caterpillar manual and calculates the grade horsepower that corresponds to the truck's required power when driving uphill in loaded mode.

$$\text{Grade Horsepower} = \frac{\text{GMW(kg)} \times \sum \text{effective grade} \times \text{Speed (km/h)}}{273.75} \qquad \text{Equation 4-9}$$

GMW: Gross Machine Weight

Effective grade: Actual grade + rolling resistance

The example is for a 789D with a nominal 1,800 hp drivetrain and a nominal payload of 200 tonnes. The truck needs 1,530 horsepower to drive on the road with an 8% slope grade and a speed of 13.2 km / h, 1530 hp is 85% of the truck's gross engine power (Caterpillar, The 48th Caterpillar® Performance Handbook, 2018).

GMW: 317520 kg

Rolling resistance: 2 %
Actual grade: 8%
Operational speed 13.2 km/h

$$\text{Grade Horsepower} = \frac{317{,}520 \times \ (0.02 \ + \ 0.08\,) \times 13.2}{273.75} = 1530 \text{ hp}$$

$$\frac{1530 \text{ grade horsepower}}{1800 \text{ gross engine hp}} * 100 = 85\% \text{ power train efficiency}$$

The same approach for Caterpillar 785D, required energy for driving loaded, 10 % slope grade with 10 km/h speed, is 1218 hp.

$$\text{Grade Horsepower} = \frac{256{,}500 \times (0.10 + 0.03) \times 10}{273.75} = 1218 \text{ hp}$$

$$\frac{1280 \times 0.74 \text{ grade horsepower}}{1005 \text{ gross engine hp}} \times 100 = 90\% \text{ power train efficiency}$$

Fuel consumption is:

$$\text{FC} = 1218 \times 0.74 \times 0.25 = 225 \text{ l/h}$$

The retarding horsepower formula calculates the energy needed to decelerate the truck on downhill roads. This time, rolling resistance is favourable for operation, so it should be reduced from the road's slope grade. While trucks can travel a little more than 30 km/h, they usually are limited to 30 km for safety reasons.

$$\text{Retarding Horsepower} = \frac{\text{GMW(kg)} \times \sum \text{effective grade} \times \text{Speed (km/h)}}{273.75} \qquad \text{Equation 4-10}$$

$$\text{Retarding hp} = \frac{116{,}500 \times (0.10 - 0.03) \times 30}{273.75} = 893.7 \text{ hp}$$

Therefore, the fuel consumption rate is:

$$\text{FC} = 893.7 \times 0.74 \times 0.25 = 165.3 \text{ l/h}$$

Dividing the rate by 30 km/h speed, 5.5 liters of diesel consumes per each kilometer.

$$\text{FC Unloaded} = 165.3 \div 30 = 5.5 \text{ l/km}$$

As mentioned above, the energy required is independent of velocity. In order to indicate this, the parameter X is assumed for the amount of velocity in equations.

$$\text{Grade hp} = \frac{256{,}500(\text{kg}) \times (0.10+0.03) \times X\,(\frac{\text{km}}{\text{h}})}{273.75} = 121.8\,X\ \text{hp} \qquad \text{Equation 4-11}$$

$$\text{FC} = 121.8\,X \times 0.74 \times 0.25 = 22.5\,X\ \text{l/h}$$

$$\text{FC} = 22.5\,X\ \text{l/h} \div X\ \text{km/h} = 22.5\ \text{l/km}$$

Therefore, fuel consumption is a function of slope grade and gross machine weight for loaded mode and machine weight, dead load, in unloaded mode. Please note that the 0.25 ratio is valid for the trucks with payloads more than 90t, and all these trucks have a similar payload ratio of 55 %. This means that regardless of the type of trucks, The formula above is able to estimate the fuel consumption for transporting material with rigid frame dump trucks based on caterpillar data and configurations, which was shown could be justified and used for other manufacturers.

$$\Sigma\text{FC}(\frac{\text{L}}{\text{km}}) = \frac{\{\text{GMW(kg)}\times(0.10+0.03)\}+\{\text{Dead Load (kg)}\times(0.10-0.03)\}}{273.75} \times 0.74 \times 0.25 \qquad \text{Equation 4-12}$$

Please note that in equation 4-8 the needed energy calculates by multiplying power engine P and load factor, here the equations 4-12 and 4-13 formulas are correspond for calculating the needed horsepower instead.

$$\text{P}\times\text{LF}(\frac{\text{hp}}{\text{km}}) = \frac{\{\text{GMW(kg)}\times(\text{Effective grade})\}+\{\text{Dead Load (kg)}\times(\text{Effective grade})\}}{273.75}\,(\frac{\text{hp}}{\text{km}}) \qquad \text{Equation 4-13}$$

Regarding payload ratio of 55%, it is also possible to accurately estimate the needed fuel for the entire deposit at the very beginning, pre-feasibility and feasibility studies, using average distance and average slope grade. Here payload means the amount of material needed to be transported.

$$\Sigma\text{FC} = \frac{\left\{\text{Payload}\times\frac{100}{55}(\text{kg})\times(\text{Effective grade})\right\}+\left\{\text{Payload}\times\frac{45}{55}(\text{kg})\times(\text{Effective grade})\right\}}{273.75}\,0.74\times0.25\,(\frac{\text{hp}}{\text{km}})$$

Equation 4-14

$$\Sigma\text{FC} = \frac{\left\{30\times10^{9}\times\frac{100}{55}(\text{kg})\times(0.10+0.03)\right\}+\left\{30\times10^{9}\times\frac{100}{55}\times\frac{45}{100}(\text{kg})\times(0.10-0.03)\right\}}{273.75} * 0.74 \times 0.25$$

$$= \frac{\{7\times10^9\}+\{1.7\times10^9\}}{273.75} \times 0.74 \times 0.25$$

$$= 5{,}958{,}904 \text{ l/km} \times 3 \text{ km}$$

$$= 17.6 \text{ M l}$$

To be more precise, one must include total idle times. This could be done by adding 37 seconds for each loading time of the assumed shovel, and for every three times of loading, add 1 minute for repositioning of trucks and 1 minute for unloading. In the end, multiply the sum with average hourly fuel consumption when idling, depending on the Vehicle class between 10 and 15 lit/h.

4.3.4 Engines and fuels´ effects

C at equation 4-7 consists of efficiency of diesel motor which is around 40% and entire power train system including gear box, drive shaft etc, and also fuel properties. Even though overall power train efficiency for diesel engines are similar , the formula is not valid for other types of engine, for example trolley assist system in which the electricity transfers to electric motors on each wheel therefore the efficiency could be enhanced up to 95%.

Electric power train systems converting diesel to electricity, therefore they still convert chemical energy to electricity with 40% efficiency.

$$FC = P \times C \times LF$$

$$\text{Fuel Consumption rate}(\text{lit/h}) = P(\text{hp}) \times \frac{0.74 \text{ kWh}}{1 \text{ hph}} \times \frac{1 \text{ lit}}{10.96 \text{ kWh}} \times \frac{1}{EE} \times LF$$

Equation 4-15

P: engine power
EE: engine efficiency
LF: load factor

$$C = \frac{1 \text{ lit}}{10.96 \text{ kWh}} \times \frac{1}{EE}$$

for normal diesel engines EE is 40% considering 5% for all losses correspond to transferring mechanical force from engine to wheels, entire power train energy efficiency equals to 35%.

$$C = \frac{1 \text{ lit}}{10.96 \text{ kWh}} \times \frac{1}{0.35} = 0.26$$

The amount is close to the one already calculated on previous section based on performance handbook from cat.

The 10.96 ratio is a general figure regarding energy content of one litter of diesel it could be replace by any other fuel properties for example by biodiesel´s ratio which is 9.58. Replacing P×LF it is possible to calculate the needed energy for trucks independent from engine and fuel type.

$$\Sigma\ \mathrm{EC}\left(\frac{\mathrm{kWh}}{\mathrm{km}}\right) = \frac{\left\{\mathrm{Pl}\times\frac{100}{55}(\mathrm{kg})\times \mathrm{EG}\right\} + \left\{\mathrm{Pl}\times\frac{45}{55}(\mathrm{kg})\times \mathrm{EG}\right\}}{273.75}\,\frac{\mathrm{hp}}{\mathrm{km}}\times\frac{0.74\ \mathrm{kWh}}{1\ \mathrm{hph}}\times\frac{1}{\mathrm{EE}}\left(\frac{\mathrm{kWh}}{\mathrm{km}}\right) \qquad \text{Equation 4-16}$$

EC: energy consumption

Pl : payload

EG: effective grade

EE: engine efficiency

As a hypothetical example if a 785D truck from Caterpillar uses 100% electricity, assuming it uses battery or connects to power network during entire cycle, by regarding 95% efficiency for a typical electric motor then the needed energy equals to 117 kWh, or 0.86 kWh/t (for 10% slope grade).

$$\Sigma\ \mathrm{EC}\left(\frac{\mathrm{kWh}}{\mathrm{km}}\right) = \frac{\left\{140{,}000\times\frac{100}{55}(\mathrm{kg})\times 0.13\right\} + \left\{140{,}000\times\frac{45}{55}(\mathrm{kg})\times 0.07\right\}}{273.75}\,\frac{\mathrm{hp}}{\mathrm{km}}\times\frac{0.74\ \mathrm{kWh}}{1\ \mathrm{hp}}\times\frac{1}{0.95}$$

$$\Sigma\ \mathrm{EC}\left(\frac{\mathrm{kWh}}{\mathrm{km}}\right) = \frac{\{33091\} + \{8018\}}{273.75}\,\frac{\mathrm{hp}}{\mathrm{km}}\times\frac{0.74\ \mathrm{kWh}}{1\ \mathrm{hp}}\times\frac{1}{0.95} = 117\left(\frac{\mathrm{kWh}}{\mathrm{km}}\right)$$

4.4 Real Measured Data

As mentioned earlier, mining companies keep a record of the fuel consumption of the entire fleet or the total fuel consumption of each truck. Baker used Kral Volumeter to collect fuel usage data and combine it with data from local GPS and VIMS to measure fuel usage for positive slope grade for both loaded and unloaded operational modes. Fig. The investigation was carried out at a surface coal mine for CAT 785D.

Figure 4-6 Installation of Kral Volumeters to fuel tank of Caterpillar 785D truck (Kubler & Baker, 2015).

The result is figure below, collected from 584 cleaned samples, which cover 21 shifts when truck was ascending.

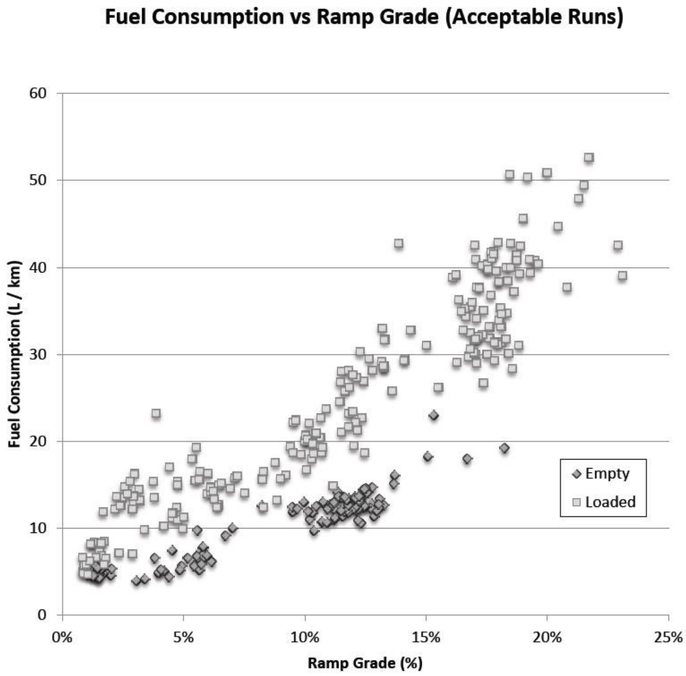


Figure 4-7 Fuel consumption rate and slope grade for CAT 785 D (Kubler & Baker, 2015).

Finally the investigation presented by two formulas correspond for loaded and unloaded modes on positive slope grades (Kubler & Baker, 2015).

$$\mathrm{f\,unloaded} = -1661\,x^3 + 727.7\,x^2 + 6.809\,x + 4.562 \qquad \text{Equation 4-17}$$

$$\mathrm{f\,loaded} = -803.8\,x^3 + 603.87\,x^2 + 80.825\,x + 7.638 \qquad \text{Equation 4-18}$$

The formulas could also be set for further benchmarking by dividing the fuel rate by the payload; however, it should be noted that the payload and fuel rate ratio is not constant. One must include load factors into account to solve this problem, the same procedure as the already outlined mathematical method.

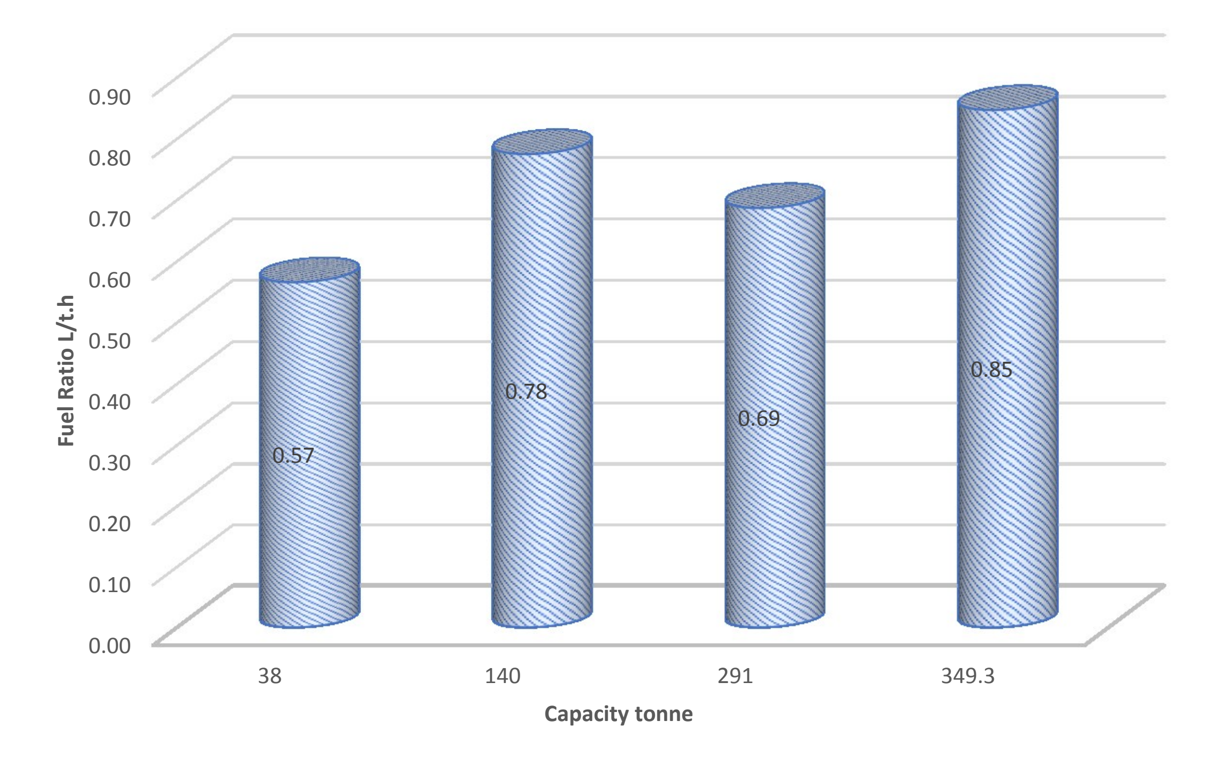


Figure 4-8 Fuel consumption rate and nominal payload ratio.

Each truck configuration has a different performance and speed for the same road profile means each class's cycle time is varied. Also, the relation between capital cost and payload amounts is not linear. Finding the optimal fleet size is a sophisticated task requiring experience and knowledge; some simulation technics like discrete event simulation have been using to dealing with these complexities.

4.5 Time-independent model

In this part, fuel consumption estimates with a time-independent approach using parameters such as vehicle mass, rolling resistance, and gradient force. to compare the mentioned systems, an index is assigned to each system configuration based on different energy sources and life cycle assessment methods used for GHG emissions estimation.

The result showed that a trolley assistance system requires a smaller amount of energy due to the better energy efficiency of electric motors compared to internal combustion engines. However, if a lignite-fired power plant supplied the electricity, the trolley assist system had more unfavorable GHG emissions than the conventional truck system. When it came to renewable sources of energy, they significantly reduced the environmental impacts.

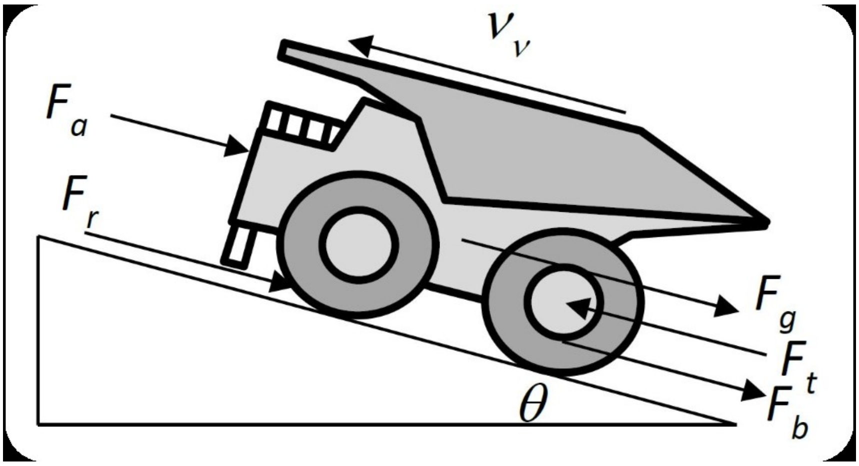


Figure 4-9 Effective Forces (Terblanche, 2018).

$m_v \ \frac{d}{dt} v_v(t) = F_t(t) - \left((F_r(t) + F_g(t) + F_a(t) + F_b(t) \right)$ Equation 4-19

m_v: the mass of vehicle

$v_v(t)$: the vehicle velocity as a function of time

$F_t(t)$: the traction force

$F_r(t)$: the rolling resistance force

$F_g(t)$: the grade force

$F_a(t)$: the aerodynamic resistance force

$F_b(t)$: the braking force

$$m_v \ \frac{d}{dt} v_v(t) = F_t(t) - \left((F_r(t) + F_g(t) + F_a(t) + F_b(t) \right)$$

For constant speed:

$\frac{d}{dt} v_v(t) = 0$

$F\ \_a\ \ (t\ )$: the aerodynamic resistance force is negligible for speeds below 60 km/h therefore the equation changes to:

$0 = F_t - \left((F_r + F_g + F_b \right) \qquad F_t = \left(F_r + F_g \right)$

F_r= rolling resistance × $m_v\ g\ \cos\theta$

θ : the slope grade

F_g= $m_v\ g\ \sin\theta$

Since the θ value is relatively small:

$\sin\theta = \theta$,and $\cos\theta = 1$

Finally for the uphill mode:

$$F_t = (F_r + F_g) = (\text{rolling resistance} \times m_v\ \text{g}) + (\text{slope grade} \times m_v\ g)$$

$$= (\text{rolling resistance} + \text{slope grade})\ m_v\ \text{g}\ [kg\frac{m}{s^2}]$$

$$F_t = (\text{rolling resistance} + \text{slope grade})\ m_v\ \text{g}\ [kg\frac{m}{s^2}]$$

$$W_{(J)} = F_{(N)} .d_{(m)}$$

$$F_t = (F_r + F_g) = 9.81\ (\text{EG.}\ m_v)$$

$$d = 1000\ \text{m} = 1\ \text{km}$$

$$1\ \text{J} = 1\ \text{Ws} = \frac{1}{3600 \times 1000}\ \text{kWh}$$

$$m_v\ \text{t} = m_v\ 1000\ \text{kg}$$

$$W_{(kWh)} = m_v \times 1000\ \frac{\text{kg}}{\text{t}} \times 9.81\frac{m^2}{s} \times \frac{\text{EG}}{100} \times 1000\ \frac{\text{km}}{\text{m}} \times \frac{1}{3600 \times 1000}\ kWh$$

$$W_{(kWh)} = m_v \times \frac{\text{EG}}{100} \times 2.752\ kWh \qquad \text{Equation 4-20}$$

$$P\ (W) = \frac{W(J)}{t\ (s)}$$

$$P\ (W) = \frac{F(N).d\ (m)}{t\ (s)} = F(kg\frac{m}{s^2}).\ V(\frac{m}{s})$$

$$P\ (W) = EG.m_v.g\ \left[kg\frac{m}{s^2}\right] \times V(\frac{m}{s})$$

$$P\ (W) = \left[EG\ .\ m_v\ V\left(\frac{km}{h}\right)\right] \times 2.725\ \left[kg\frac{m^2}{s^2} \times \frac{1}{s}\right]$$

$$1\ \frac{J}{s}\ \text{or watt} = 0.00134\ hp$$

$$P\ (hp) = \left[EG\ .\ m_v\ V\left(\frac{km}{h}\right)\right] \times 2.725\ \left[\frac{J}{s}\right] \times 0.00134\ hp$$

$$P\,(hp) = \frac{EG\,.\,m_v\,V\left(\frac{km}{h}\right)}{273.86}\,hp$$ Equation 4-21

4.6 Noise emission

Surface mining involves many mechanized processes, some of which require part-time or full-time operators or observers. Although increased automation and the use of robots in the industry are reducing the number of people in hazardous areas, the need for human intervention exposes many workers to the health risk of noise-induced hearing loss (NIHL). A prerequisite for any control efforts is to define the duration and specification of exposure based on standard measures. The time-weighted average noise level is used to determine the average noise level over a period of time in each area. The action level is the level of protection at that level, i.e., the measures required to prevent hearing damage, and the threshold is the limit of noise exposure that should not be exceeded.

Table 4-8 noise samples exceeding specified TWAs sound levels (Bauer & Kohler, 2000).

Occupation	Number of samples	% of samples > 90 dBA	% of samples > 85 dBA
Front-End-Loader Oper	12812	12.9	67.7
Truck Driver	6216	13.1	73.7
Crusher Oper	5357	19.9	65.1
Bulldozer Oper	1440	50.7	86.2
Bagger	1308	10.2	65.0
Sizing/ Washing Plant Oper	1246	13.2	59.7
Dredge/ Barge Attendant	1124	27.2	78.7
Clean-up Person	927	19.3	71.3
Dry Screen Oper	871	11.7	57.6
Utility Worker	846	12.4	60.6
Machenic	761	3.8	43.9
Supervisor/ Administrator	730	9.0	32.2
Laborer	642	17.1	65.7
Dragline Oper	583	34.0	82.5
Backhoe Oper	546	8.4	52.6
Dryer/Kiln Oper	517	10.5	55.5
Rotary Drill Oper (Electric/ hydraulic)	543	39.6	83.1
Rotary Drill Oper (Pneumatic)	489	64.4	89.0

The same approach, using a personal dosimeter, can be used to measure the noise exposure for additional truck classes operators and maintenance and repair technicians. The point is that the measurements need to be converted to TWA for 8-hour shifts or any other desired span. Below are the equations and examples for conversion of 86 dB for six hours and 92 dB for three hours over a 9-hour shift (NoiseMeters Inc, 2022).

The first step is to calculate the dose:

$$D_{\%} = 100 \text{ x } (C1/T1 + C2/T2 + C3/T3 + ... + Cn/Tn) \qquad \text{Equation 4-22}$$

Cn: Exposure time

$Tn = 8 / 2^{(L-90)/5}$

L: Sound level

Once the dose calculated, TWA can be calculated using the following equation:

$$TWA = 16.61 \text{ Log10 } (D/100) + 90 \qquad \text{Equation 4-23}$$

The example calculation:

$D = 100 \text{ x } (6/13.9 + 3/6.1) = 92.3\%$

$TWA = 16.61 \text{ x Log10 } (92.3 / 100) + 90$

$TWA = 89.4$ dB

Finally, schedules can be created for operators depending on the standard limits for each country. The table lists the NIOSH standards. For the European standards, see Table 3-1.

4.7 Dust emission

Haul trucks are responsible for the majority of dust emissions in surface mines. Almost 80 % of the particles with a size of less than 10 micrometers originate from the operation of the trucks [(Cole & Zapert, 1995) (Amponsah-Dacosta & Annegarn, 1998) (Reed, Westman, & Haycocks, 2001)]. Although respirable dust accounts for nearly 20% (Organiscak JA, 2004), it is still important to control the dust to avoid accidents due to poor visibility. If the dust contains respirable crystalline silica, workers are at risk of overexposure and lung diseases. Dust samples exceeding the applicable or reduced respirable dust standard due to the presence of silica were 12% in sand and gravel mining, 13% in stone mining, 18% in nonmetal mining, and 21% in metal mining (Organiscak, 2010). Road construction and maintenance, speed and traffic controls, and water spraying are the most effective and common dust control practices for haul trucks (Cecala & al, 2019).

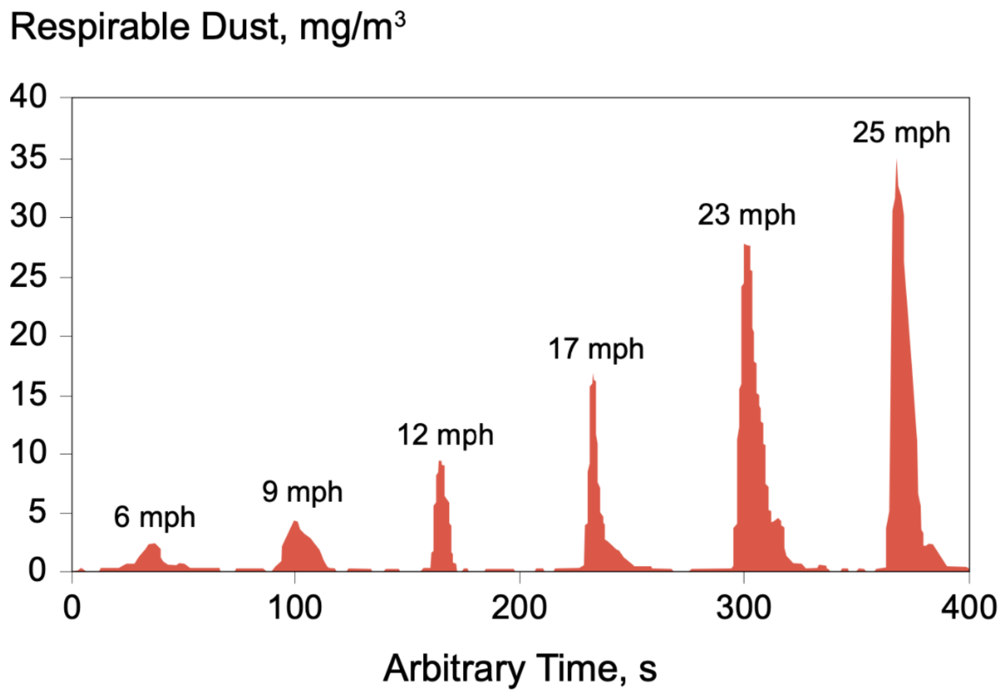


Figure 4-10 dust profile measured at roadside for a haul truck (Cecala & al, 2019) data adopted from (Thompson & Visser, 2001).

The figure above indicates the speed of the trucks and the dust emissions in the air. Limiting the speed of trucks and controlling the distance between them can reduce dust emissions, though it can not be considered a sound solution since the production rate will be compromised. On the other hand, wet spraying is a cost-effective and practical measure. The spraying schedule must be planned depending on the road materials, the speed of the trucks, and the weather conditions.

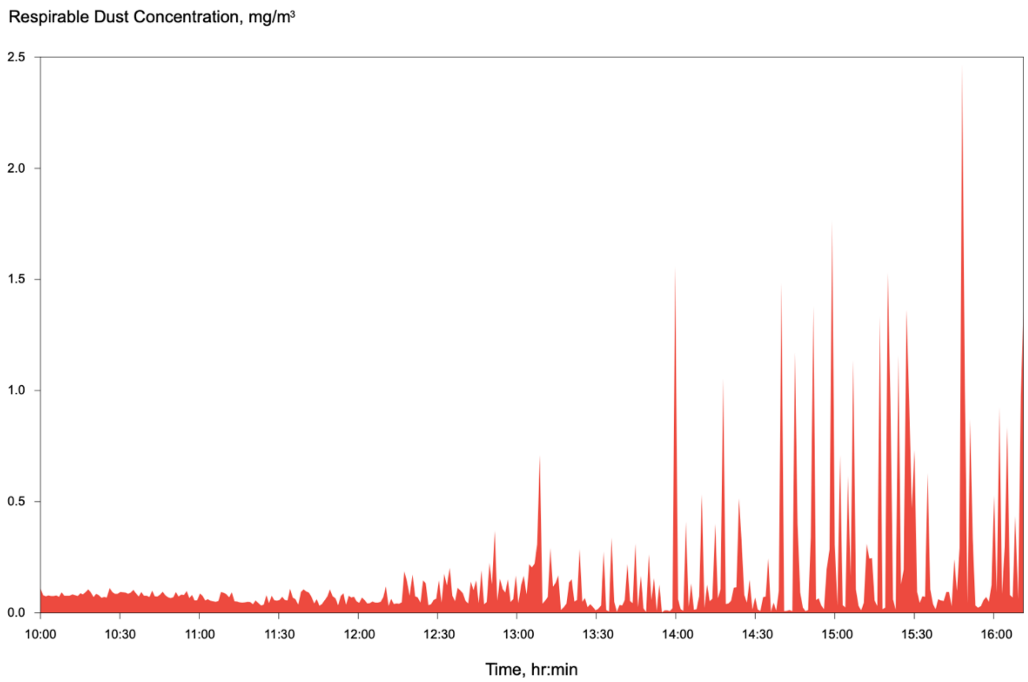


Figure 4-11 respirable dust concentrations measured from a haul road after water application occurred at 10:00 (Organiscak JA, 2004).

5 Results and Comparison

5.1 Introduction

Accurate assessment of choices depends on the representativeness of evaluation techniques that can measure one or more attributes identically. When possible, quantifying the attributes gives a concrete picture and makes the comparison more explicit. However, to avoid misconceptions, it is critical to understand what is being left behind or ignored in the numbers. Fundamentally, assumptions are about compromises, and this had to be kept in mind during calculations and later when comparing the final results.

Transport systems, as a case in point, have distinct characteristics and ways of operation. The function is the same, namely the transport of material, but otherwise, trucks and conveyors are quite unlike machinery sets. The comparison of noise exposure presented in chapters three and four is quite representative for a truck driver because it is measured with a personal dosimeter, whereas for a conveyor belt, it is not so easy to determine the location of the supervisor and the exposure duration. It is assumed that the supervisor at the conveyor maintains a certain distance from the belt and the transport points, which is not always the case and partially affects the perception of actual exposure factors.

5.2 Challenges of comparative method

The fact that dust, noise, and vibration studies and techniques are conducted for safety reasons, and the essence of the assessment is to get a picture and analyze personnel safety. While the ecological assessments are conducted primarily to obtain or justify environmental licenses and credentials, they focus on long-term issues such as greenhouse gas emissions, underground water regimes, and changes in the landscape. The approach is authentic primarily because environmental impacts are considered long-term rather than temporary in terms of sustainable development. For example, environmentally oriented approaches consider the current and future conditions of residents in the mining area in addition to the general social concerns of the public, while safety approaches are concerned with minimizing work-related health problems of miners.

Dust, noise, and vibration are highly case-specific, and mitigation measures and practices vary from site to site. Therefore, it is difficult to say how much noise, vibration, or dust per ton of material is emitted for each system configuration. On the other hand, energy efficiency and greenhouse gas emissions can be estimated for each ton of material quite independently of the individual case details. In this chapter, energy efficiency and greenhouse gas emissions for transport configurations are compared based on practical characteristics. A calculation model was developed that includes the equations and formulas for both systems to facilitate the comparison of energy consumption and GHG emissions for possible material handling options.

5.3 General assumptions

- The model does not consider the fuel consumption during loading, unloading, and standing in lines. One can add 5 % to overall fuel consumption to estimate those idle time's fuel consumption.
- As there is no idle time, the use of a dispatch system or any fleet management system was not assumed. In other words, the tool assumes trucks are loaded and unloaded instantly without any kind of delay. Therefore there is no room for truck allocation optimization, and logically, trucks came back empty from a discharge location using the same route, which they already used to get there. However this approach underestimates the truck's fuel consumption, making it possible to be compared with a belt conveyor.
- The results are valid for rigid frame off-highway trucks that have a payload larger than 100 metric tons.
- The ratio between payload and dead load of trucks is assumed to be 55%. The weight of a truck which can provide nominal payload of 100 ton is 81 tons.
- The fill factor of the trucks assumes 100%.
- Trucks operate with constant speed without acceleration or de-acceleration. There is no traffic light or changing direction, which needed the trucks' speed to be reduced.
- The effect of the operator's performance does not take into account.
- The truck's performance will not reduce over time, always regarded as brand new vehicles.
- Calculations for belt conveyor are based on RWE's belt conveyor with 17500 t/h production, 1500 m length, and 2200 mm width.
- All units are metric.

5.4 Parameters of model

Slope grade [%]: use a positive value for slope grade; this means the loading point(s) has a lower altitude than the discharge point(s). The tool automatically takes the negative amount into account for empty trucks when coming back downhill.

Mass of Material [t]: the model is valid for estimating required energy for options below:

- One ton of material, simply put "1" for the Mass of materials cell and define your desire road profile.
- A truck, use truck´s actual payload (97% of nominal payload).
- A yearly production of a mine, put production per year and overall average transporting distance and slope grade.
- A deposit, put the entire amount of a mine´s reserve (plus waste amount) and average slope and distance.

Rolling resistance [%]: a reasonable guess would be a value between 2 and 3. Please see the table below for other conditions.

Table 5-1 Rolling resistance for different road conditions (Caterpillar handbook, 2018).

Road Condition	Rolling Resistance (%)
Hard, well-maintained, permanent haul road	1.2
Well-maintained road with little flexing	2.5
Road with 25 mm (1 in) tire penetration	4
Road with 50 mm (2 in) tire penetration	5
Road with 100 mm (4 in) tire penetration	8
Road with 200 mm (8 in) tire penetration	14

Distance [km]: similar to the mass of material, the distance could be a specific length of a road or an average for a mine.

Type of Fuel: the second sheet of the excel file corresponds to fuel descriptions, the references are stated in comment notes, and they can be updated easily. Besides, there is an option for custom fuel, which you can justify as you wish.

Engine Efficiency [%]: this is a ratio for combustion engine efficiency.

Source of Electricity: The source of electricity defines Green House Gases' emission for each unit of energy (CO2 e/kWh). The values could be adjusted using the electricity's sheet.

Recuperation System [%]: refers to trucks with an electrical power train system that can generate and save electrical energy when driving downhill roads. The portion defines the part of a road that system could save energy from. For example, if we had a 3km road and the recuperation system engage 2km, then the portion value would be 66%. Also, it was assumed the efficiency for saving electrical energy from mechanical energy is 85%.

Trolley Assist System [%]: the percentage defines the portion of a road on which trucks are able to be connected to a power network and consume electricity instead of liquid fuels.

5.5 Types of Systems

Combustion engine: refers to normal trucks consume liquid fuels.

Equation for uphill mode:

$$RE = \frac{\left\{Mass\ of\ material \times \frac{100}{55}(kg) \times (Slope\ grade + Rolling\ resistance)\right\}}{273.75}\ hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1\ lit}{9.96\ kWh} \times \frac{1}{EE} \times D$$

Equation 5-1

RE: required energy

EE: engine efficiency [%]

D: is Distance [km]

1 hp = 0.74 kWh

Equation for downhill mode:

Equation for downhill mode:

$$RE = \frac{\{Mass\ of\ material \times \frac{45}{55}(kg) \times (Slope\ grade - Rolling\ resistance)\}}{273.75} hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1\ lit}{9.96\ kWh} \times \frac{1}{EE} \times$$

D Equation 5-2

Combustion engine equivalent: converts the required energy to kWh in order to make comparison more practical.

Equation for uphill mode:

$$RE = \frac{\{Mass\ of\ material \times \frac{100}{55}(kg) \times (Slope\ grade + Rolling\ resistance)\}}{273.75} hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1}{0.40} \times D$$

Equation 5-3

Equation for downhill mode:

$$RE = \frac{\{Mass\ of\ material \times \frac{45}{55}(kg) \times (Slope\ grade - Rolling\ resistance)\}}{273.75} hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1}{0.40} \times D$$

Equation 5-4

Theoretical required energy: the minimum required energy regardless of systems´ types. The Efficiency factor must be 100 % or could be eliminated from equations 5-3 and 5-4.

Electric power system: calculates the required energy for a hypothetical truck which consumes 100% electricity.

Equation for uphill mode:

$$RE = \frac{\{Mass\ of\ material \times \frac{100}{55}(kg) \times (Slope\ grade + Rolling\ resistance)\}}{273.75} hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1}{0.95} \times D$$

Equation 5-5

Equation for downhill mode:

$$RE = \frac{\{Mass\ of\ material \times \frac{45}{55}(kg) \times (Slope\ grade - Rolling\ resistance)\}}{273.75} hp \times \frac{0.74\ kWh}{1\ hp} \times \frac{1}{0.95} \times D$$

Equation 5-6

5.6 Energy Consumption

The diagram below shows the energy required to transport one ton of material over a distance of one kilometer. Two methods for truck energy efficiency assume that the conventional method is based on the original equipment manufacturer's fuel consumption value and that the measured data is taken from an opencast coal mine in Australia. Both results are for a conventional diesel truck with a 140-ton payload. The data for the conveyor belts refer to the energy consumption of conveyor belts with different inclinations, the width of all belts is 2800 mm, the experimental data is from Hambach, and the calculated data is based on the German DIN standard.

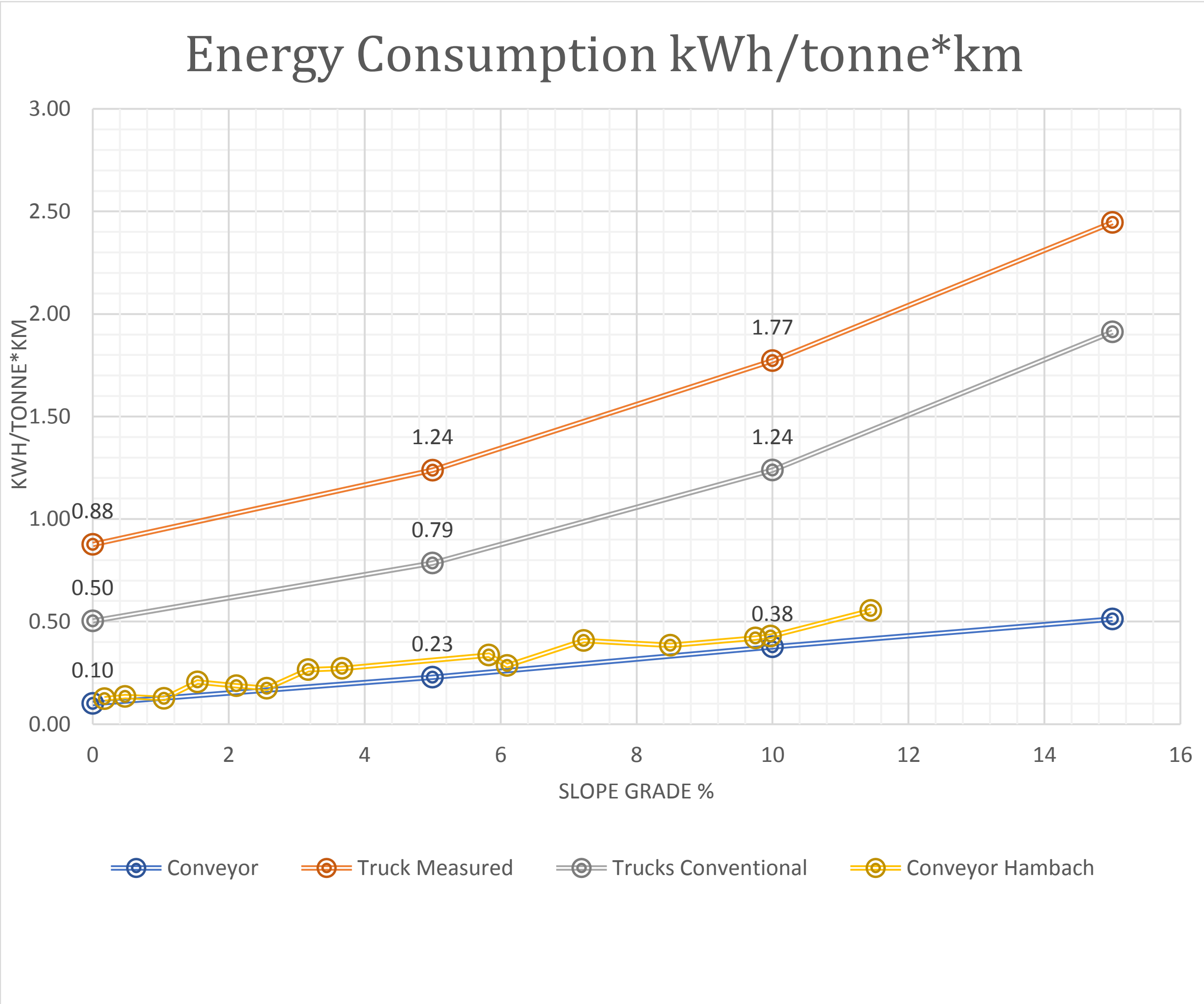


Figure 5-1 Based on Cat 785 D and Hambach mine 2800 mm conveyors data (18000 t/h).

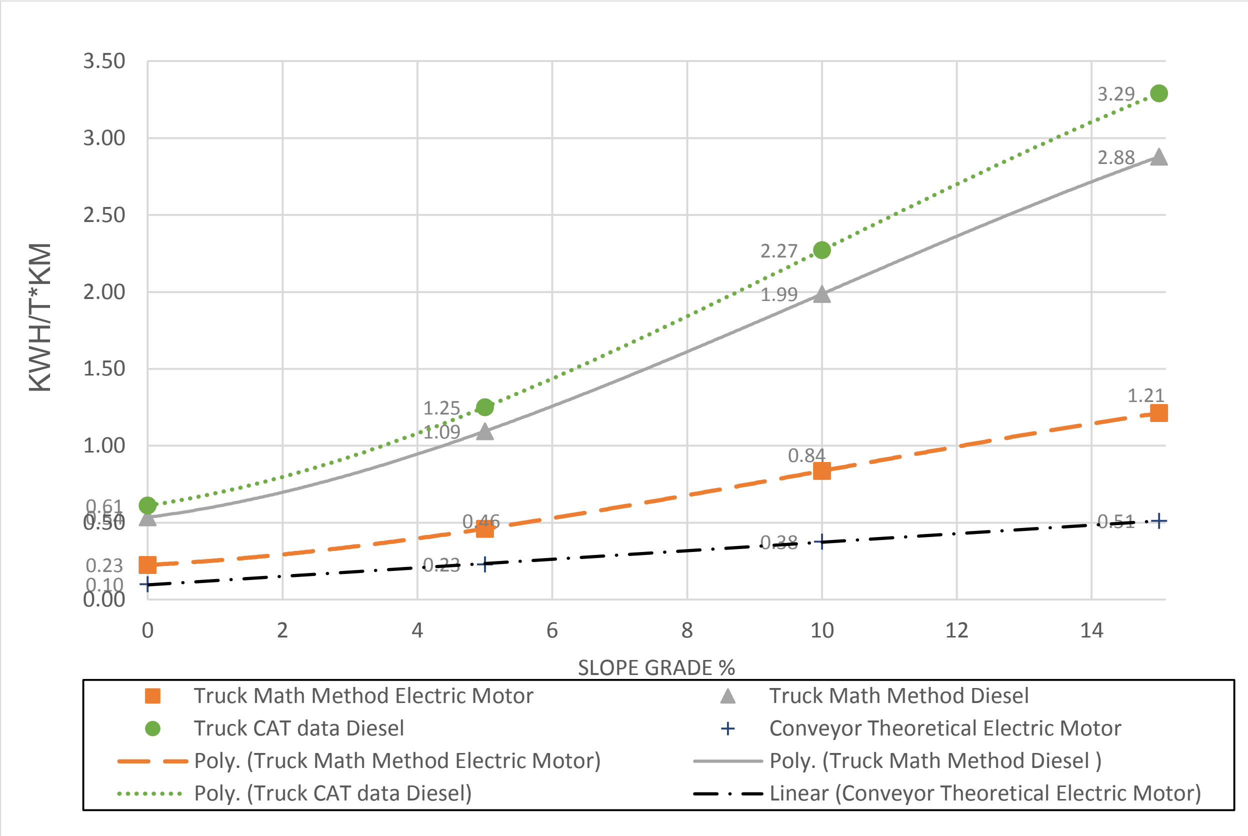


Figure 5-2 Energy efficiency for haul truck with electric and diesel engine.

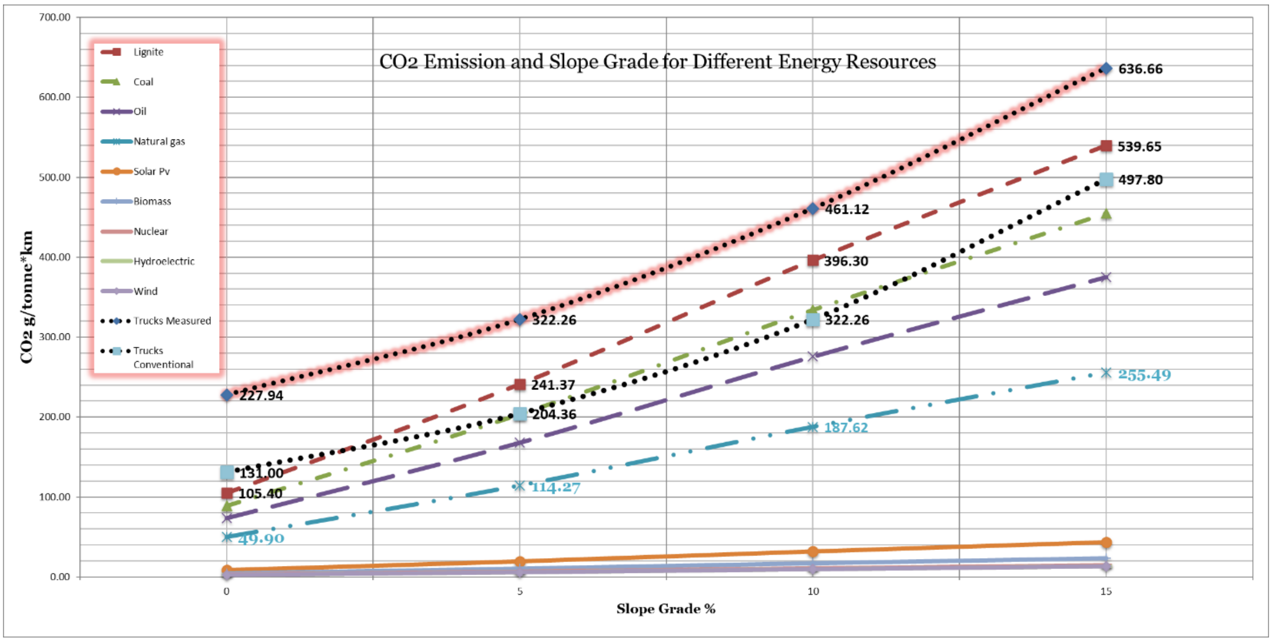


Figure 5-3 Based on Cat 785 D and Hambach mine 2800 mm conveyors data (18000 t/hr).

5.7 GHG emissions comparison for two case studies

To compare greenhouse gas emissions, both the amount and source of energy must be defined. Theoretically, the energy required to transport a given amount of material is constant; however, each system has its own efficiency, so the energy consumed varies. For example, the efficiency of an internal combustion engine is about 40 percent, which means that most of the energy contained in the diesel is wasted, while the electric motor efficiency is about 95 percent. The other factor is rolling resistance. In the case of conveyors, it represents the resistance force of the rollers; in the case of trucks, it is the resistance force induced by the penetration of the tires into the road. The tables and diagrams below are the model results for two theoretical production rates of 30 and 100 million tonnes.

Table 5-2 Energy consumption and GHG emissions 30 m tonnes production rate.

Production: 30 m tonnes	Tranportation distance: 5 km		Slope grade: 8%	
	Energy m KWh	**CO2 e t**		
Electricity Source		**Wind**	**Germany avg**	**Lignite**
Diesel	**243**	81.111	81.111	81.111
Diesel + recuparation	**187**	62.469	62.469	62.469
Trolley assisst	**163**	36.434	58.249	96.238
Trolley assisst + recuperation	**94**	13.552	35.367	73.355
Conveyor	**49**	1.296	19.990	52.544

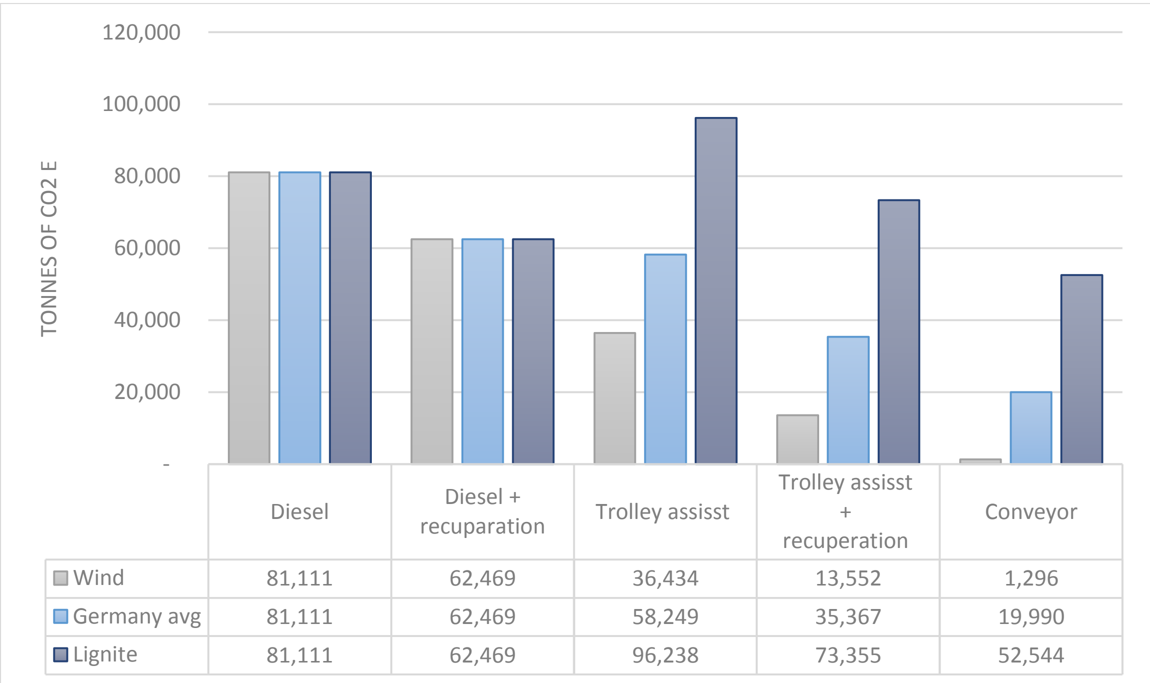


Figure 5-4 CO2 e comparison for different transportation systems 30 m tonnes production rate.

Table 5-3 Energy consumption and GHG emissions 100 m tonnes production rate.

Production: 100 m tonnes	Tranportation distance: 10 km		Slope grade: 8%	
	Energy m KWh	**CO2 e t**		
Electircity Source		Wind	Germany avg	Lignite
Diesel	1.617	540.740	540.740	540.740
Diesel + recuparation	1.246	416.426	416.426	416.426
Trolley assisst	1.084	242.899	388.333	641.584
Trolley assisst + recuperation	628	90.345	235.781	489.033
Conveyor	330	8.640	133.270	350.290

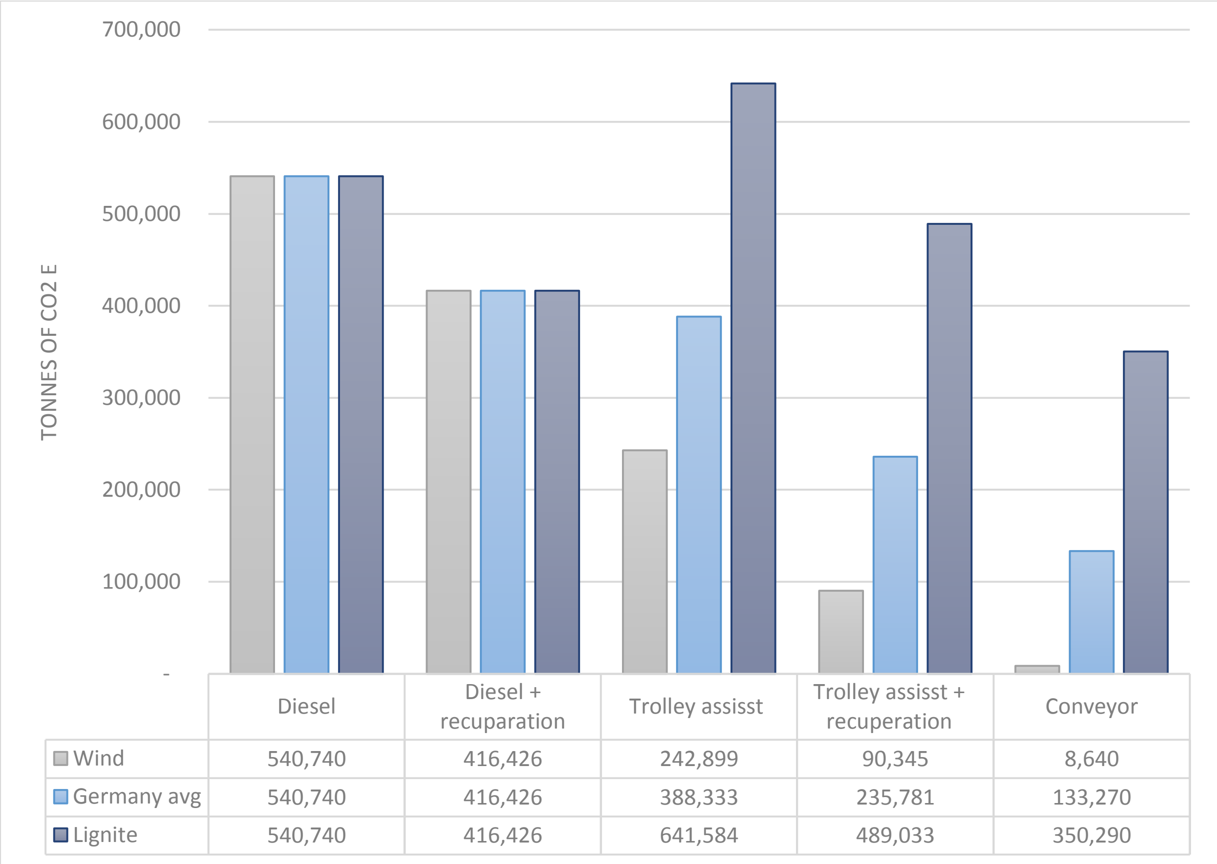


Figure 5-5 CO2 e comparison for different transportation systems 100 m tonnes production rate.

5.8 Cost analysis

To compare the practicality of conveyor and transport systems, it is important to compare both capital and operating costs. The key factor for equipment selection and cost analysis is the production rate. To properly understand the impact of production rate, capital and total costs are calculated for ten different production rates.

Despite the high investment cost of the production system, the total cost is relatively low, especially for deposits with a higher production rate. This means that the cost per tonne for the trucking system roughly fluctuates around \$2 per tonne, while the cost of the production system decreases as the production rate increases.

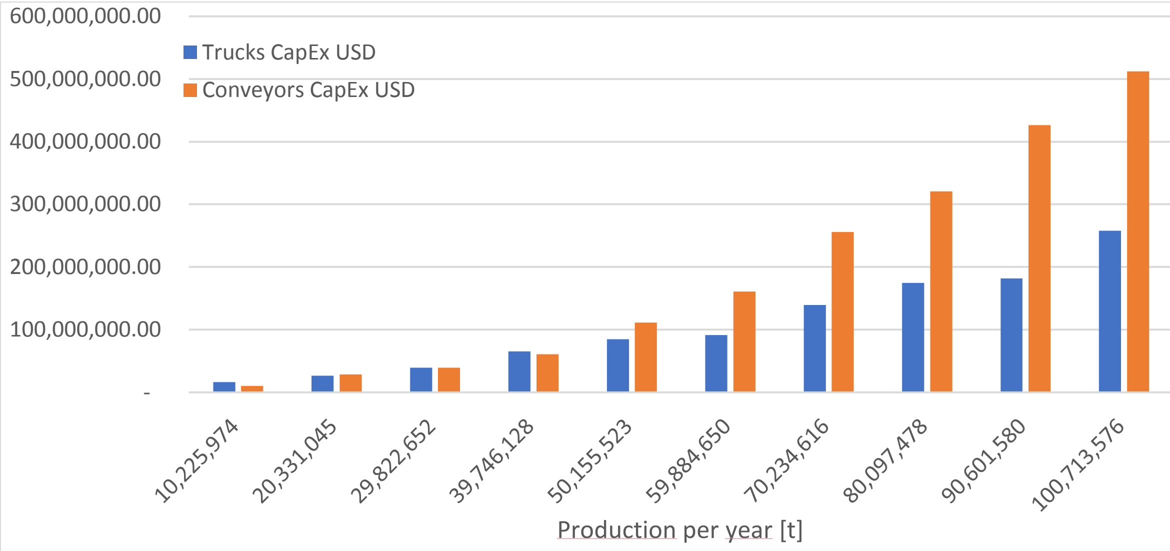
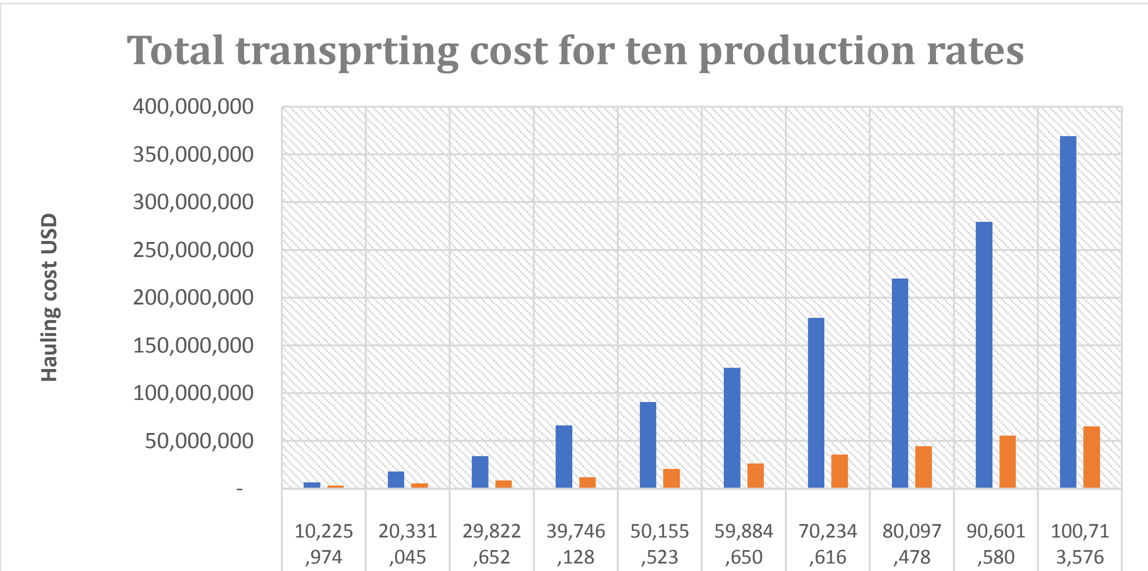


Figure 5-6: Trucks and conveyors capital cost for different production per year rates (Leinart, 2010).



	10,225 ,974	20,331 ,045	29,822 ,652	39,746 ,128	50,155 ,523	59,884 ,650	70,234 ,616	80,097 ,478	90,601 ,580	100,71 3,576
Trucks hauling cost	6,438,1	17,923,	34,192,	66,353,	90,772,	126,650	178,812	219,906	279,362	369,063
Conveyors haluing cost	3,322,7	5,661,0	8,419,2	11,925,	20,524,	26,438,	35,655,	44,469,	55,533,	65,087,

Figure 5-7 Trucks and conveyors total cost for different production per year rates.

6 Conclusion

Mining is a capital-intensive industry; although a limited number of mines operate less by market mechanisms to extract strategic minerals, the vast majority of mine sites must be explored, developed and rehabilitated according to financial incentives. This means that relationships between sectors, both within mines and throughout the supply chain, are regulated and almost constrained by financial conditions. As a result, all practices and activities are part of business strategies through their costs. If the costs of the practices were not high, the importance of the participation factor in the management and planning process would logically be lower. This is the case with GHG emissions in the mining sector, where the price of fuel and energy is relatively low compared to other costs.

Increasing global warming emphasizes the need for energy management for more than just fuel cost savings. Whether environmental legislation, such as the carbon tax, could redefine the importance of energy management for strategic decision-making in the industry depends on an assessment of potential operational choices. This study has successfully quantified a partially descriptive comparison of GHG emissions from transportation systems, which helps to some extent to evaluate and compare the two common methods. IPCC systems have better energy efficiency than current truck system configurations, including the trolly assist system. Since the conveying systems are electrically powered, they are already available to use renewable energy sources. At the same time, trucks rely on fossil fuels in whole or in part, depending on which system is used to power them. The comparative model has shown that for a sustainable transportation system, locating crushers near the loading areas is a more environmentally friendly and economical solution.

Sustainable management encompasses and considers fundamental elements and aspects of the business. Although quantifying the advantages and disadvantages of each activity is essential for sound assessment and rational decision-making, it is not enough. Pursuing only cost-cutting approaches or purely financial strategies could lead to unsustainable development that creates more challenges in the medium or long term. These challenges could be limited to one region or country and threaten the sustainable development of societies and ecosystems, such as artisanal mines in Central Africa and the forests of the Amazon, or they could have a global impact on the democracy and planet's climate. As modern mining evolves, a modern mindset is needed to create a sustainable management culture that secures the life and future of humanity.

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8 Annex